

Innovations in uranium exploration, mining and processing techniques, and new exploration target areas

*Proceedings of a Technical Committee meeting
held in Vienna, 5–8 December 1994*



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INNOVATIONS IN URANIUM EXPLORATION,
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AND NEW EXPLORATION TARGET AREAS

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FOREWORD

Despite difficulties on the part of a portion of the public to accept nuclear power, uranium continues to be an important energy resource. In 1994 there were 432 nuclear power plants in operation with a combined electricity generating capacity of 340 347 MWe. To achieve this, 58 000 tonnes of uranium were required as nuclear fuel.

In view of its economic importance, the International Atomic Energy Agency has had a long-standing interest in uranium exploration, resources, production and demand. This is reflected in numerous publications covering different aspects of this field. Particularly worth mentioning is the periodical "Uranium Resources, Production and Demand", published jointly with the Nuclear Energy Agency of the OECD. Its fourteenth edition was published in early 1994.

It was the objective of this Technical Committee meeting, the proceedings of which are presented in this TECDOC, to bring together specialists in the field and to collect information on new developments in exploration, mining techniques and innovative methods of processing that are more environmentally friendly.

The meeting was attended by a total of 22 participants from 14 countries. Eleven papers were presented describing new exploration areas, improvements in processing methods, new mining techniques for the extraction of high grade ore, and innovative approaches for site reclamation. Two working groups were organized and dealt with the analysis of world uranium resources and the new direction of research in mining and ore processing.

The meeting showed that great progress is being made in terms of efficiency of mining operations, protection of workers and the environment.

The participation and contributions made at the meeting are gratefully acknowledged. Thanks are also extended to the session chairman, H. Barthel from the Bundesanstalt für Geowissenschaften und Rohstoffe in Germany and J-L. Narcy from Cogema Resources Inc. in Canada.

The IAEA staff member responsible for the organization and implementation of the meeting was J-P. Nicolet from the Division of Nuclear Fuel Cycle and Waste Management.

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SUMMARY OF THE TECHNICAL COMMITTEE MEETING

The objective of the Technical Committee Meeting on Innovations in Uranium Exploration, Mining and Processing Techniques and New Exploration Target Areas was to discuss the status of and new trends in exploration, improvements in milling methods, mill design and tailings disposal, new research in mining methods and exploration status and supply-demand.

Since about 1980, the international uranium market has been experiencing a decline, mainly as a consequence of the slower than expected growth in nuclear power development in the world. This decline has affected all areas of the uranium industry, ranging from exploration to production and has also had an impact on employment and the financial standing of the mining companies. This decline has also resulted in the consolidation of the uranium industry through merger and acquisition and the closing down of operations that were unable to survive a low price market. At the present time only companies with low operating costs and countries that have a national strategy for their nuclear programmes are active in exploration.

For the past few years the demand for uranium has been met by drawing on various stockpiles that provided about 45% of the requirements of nuclear power plants. The supplies coming from mines represent only 55% of the total requirements. If the supplies available from stockpiles decrease, as is expected, there will be an acute shortage until mining companies can increase production to meet the requirement. This shortage may be less acute when the HEU stockpile becomes available.

Exploration funds are now mainly utilized for detailed exploration delineation and development of orebodies. Industry and government expenditure in the Member States have been estimated to be US \$79 350 000 compared with US \$164 672 000 in 1989. If this trend continues, it will be difficult to find major deposits to replace those being exploited.

The papers related to new exploration target areas indicated that nonconformity type deposits may exist in areas aside from those already known in Canada and Australia. The paper presented by R.M. Sinha from India gave such a perspective. The paper presented by A.V. Tarkhanov from the Russian Federation also indicated potential for in situ leaching operations.

Three papers from the Czech Republic describe the state of certain in situ leach mining operations and the reclamation work being carried out. Of particular interest is the paper by V. Stojek on induction log as a good watchdog which provides a monitoring method to survey the flow of a polluted groundwater mass.

The paper presented by S.P. Collings of the United States of America on the Crow Butte ISL Project describes a state of the art operation in ISL techniques with emphasis on environmentally friendly methods. The description of aquifer restoration and decommissioning provides a fine example of good mining practice.

High grade deposits present a major challenge in terms of radon emanation and radioprotection. Underground mining of high grade ore presents even greater difficulties as it is not possible to have workers in contact with the ore. Several mining methods have been considered to solve these difficulties and one of the newest methods is the jet boring method being tried at the Cigar Lake deposit in Saskatchewan, Canada. This method is based on the capability of a high pressure water jet to desegregate the ore into a slurry that is being transported through pipes to the mill. Prior to its disintegration by jet boring the surrounding ground and the ore is frozen, ensuring stability of the weather rocks and controlling water inflows and radon emanation during development and mining.

The jet boring method is a new entry mining method allowing for the mining of cavities away from drift located below the orebody. Workers are separated from the ore by a thick layer of barren rock which shields and protects effectively from gamma exposure. This method, however, is not a universal method as several criteria need to be considered to determine its applicability.

Although most of the uranium production comes from high grade mines and in situ leaching operations, the conventional mining operation with grades much lower than the high grade mines have improved their methods of mining and milling.

The paper on application of radiometric ore sorting to Kalimantan ores (Indonesia) illustrates well the success of the improvement of a milling method. Very low grade ore can be pre-concentrated at low cost to obtain a grade that can be treated with conventional extracting methods. This same method will also allow uranium recovery to be linked to rare earths. For the success of this method certain criteria must be met such as a sufficient heterogeneity of uranium minerals among ore particles, a hard rock producing few fines through crushing and a constant radiometric equilibrium between uranium and radium in the ore.

Research on the counter-current heap leaching process for treating uranium ores with common grade in China using bacterial heap leaching can shorten the heap leaching period and increase the uranium recovery rate.

Discussion among the participants indicated that progress and innovations are being made in the protection of the environment through improved flow modelling for ISL operations and control of water tables, and also through early collection of baseline data for better mitigation. Hydraulic hoisting of high grade ore slurry and 'non-entry' mining methods allow better protection of the worker.

Research into the design of mills is also being done where the mills are divided into two parts: a lower floor where high grade ore is first treated and an upper floor where radiation is minimal.

The burying of tailings in pits with pervious surroundings and a clay cover at the end of the operation is the safest method to dispose of tailings.

The participants also acknowledged that at the present time there seem to be sufficient resources to meet demand. However, as not many new mines are being brought to production and high grade mines may not always be 'low cost' due to the difficulties in extracting high grade ore, it may be difficult in the future to continue to have access to cheap supplies.

URANIUM EXPLORATION AND THE ENVIRONMENTAL PROTECTION ISSUE

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Abstract

Exploration activities are nowadays the critical basis for energy production in the future. Exploration, however, has become more and more handicapped by growing environmental concern and public policies. This impact, different as it is in the various countries, not only affects exploration efforts but particularly the international competitiveness of companies in today's changing world of mineral and energy markets. In order to master this situation and to comply with modern environmental consciousness, explorers have to embark on a positive environment protection policy but at the same time they have to stand up for their justified cause against unreasonable demands by legislators or irrational interventions by pressure groups.

Mining and in consequence exploration belong to the oldest professions of mankind and remained so until less than a generation ago — practically until the 1970s in the industrialized western countries.

Environmental sensitivity of the public has meanwhile not only resulted in a negative attitude towards mining but also in intervention either directly through pressure groups or through legislation. And if environmental activists of our "modern" society had the saying, mining and exploration would be outlawed.

Environmental idealists pursue the theory that mining disturbs unnecessarily nature since most commodities for modern civilization can be recovered by recycling of existing waste materials — not to mention reduction of consumption. Since this attitude can only lead to the ruin of mankind, there should be no tendency in the exploration and mining society to yield to the unrealistic demands of any activists who essentially only abuse or even want to eliminate our democratic system.

On the other hand, public sensitivity for environmental protection should not be underestimated any more. And there is some reason for this. Quite a few exploration crews left a mess behind, garbage piles, rotten camp grounds or contaminated drill hole or excavation residues, when they left the project — a behaviour that was and is not tolerable!

Environment related laws, decrees or ordinances affecting exploration and mining operations, and decommissioning of mine and rehabilitation of exploration sites as well have been and will be established by regulators and governmental authorities. As necessary as they are, in quite a few instances these regulations are overlapping, contradictory, unpractical, or without the appropriate perspective for reality. In part the mining society bears its share for such deficiencies due to its ignoring of the regulatory process instead of getting actively involved in the drafting of bills with praxis oriented proposals as for example in Canada.

In consequence, not only that exploration has almost arrived at a no-profession, those still in business encounter in their practical work a great burden of inconvenience and time consuming engagement due to the direct or indirect interference of pressure groups and confrontation with a flood of cumbersome but not necessarily meaningful regulations governing not only the technical conditions for exploration as it was in the past but also the environment protection requirements for any exploration activity, and related exposition to public inquiries and media interest.

So far in brief the present public and regulatory environment affecting exploration. But what is the answer to the challenge by modern environmental sensitivity and interest of the general public?

How can the explorationist counter unreasonable requirements and at the same time contribute to rational, realistic measures for the protection of the environment?

To avoid any misunderstanding, rational environment protection is an absolute necessity, too important to leave it to hypocrites. For any serious individual working in and with nature, as it is the case for explorationists, it must be a fundamental rule to preserve the environment. But this cannot mean that man is not anymore permitted to use the — for mankind necessary — wealth of nature and to give in to unreasonable demands or restrictions.

What can the explorationist do?

A number of basic rules and precautionary measures may be considered for adequate conduct of work and the tackling of environment related issues, particularly to deal with the challenge for reasonable and unreasonable demands or conditions. Here are some selected suggestions:

- Project planning should include precautionary measures for avoiding unnecessary environmental damage, e.g. by proper casing and later solid plugging of drill holes particularly of those holes intersecting aquicludes separating different groundwater horizons, and likewise it should provide for restoration of disturbed or contaminated sites such as drillsludge pits, leakage of chemical fluids from drill sites, dumps at adits or shafts sunk for underground investigations, and others.
- Familiarization with all regulations, directives etc. governing environmental issues should become a standard in order to avoid later trouble and extra costs. This task includes a sober assessment of all real or potential applicable regulations with respect to their validity for the work to be carried out, or in case of conflict or ambiguity, for negotiations with the competent authorities in order to achieve a compromise for compliance with relevant regulations.
- Communication with legislators on state, province and local community levels should be sought to transfer the industry's view as input to law making. This may help to draft from the early beginning realistic and durable laws which are beneficial for the society as a whole and the mining people as well.
- Communication with all national and local mining, environmental and other competent government authorities involved in commissioning, control, decommissioning and remedial requirements should be maintained to keep informed on all regulatory developments.
- When working is planned on private or tribal land, pre-exploration negotiations should try to find early agreements with landowners, particularly if native tribes are involved, to avoid conflict with real or pretended religious feelings and traditional customs or by unintentional profanation of sacred sites, and to compensate for interference with their daily work, loss or damage of land, vegetation and water. Compensation may include employment of locals, or special contributions or offering a share in the venture to the community. For example, in spite of all the opposition of the aboriginals in the Northern Territory of Australia or Indian tribes in Canada and the USA against uranium mining in the early 1980s, attractive financial offers or other fringe benefits by the mining companies to the local tribes were mostly more effective than complaining about the oppositional society.
- Information of the general public should be as comprehensive as possible. Voluntary information — and education — of the public either directly through public hearings or on-site demonstrations of working procedures and cleanup efforts or indirectly through the media should become an integral part of any exploration programme. — Admittedly, an awesome and often frustrating endeavour.

- Gathering of baseline environmental data should become a part of any exploration programme in order to supply the critical environmental information required for environment impact statements (EIS) and environment impact assessments (EIA) as part of the commissioning process for mining projects. Data to be collected include geological, geochemical, hydrogeological, hydrological, seismicity, geographic, physiographic, climatic, meteorological, atmospheric, aquatic, marine and terrestrial ecological, biological, archeological, historical, cultural, demographic and socio-economic information (for details see relevant issues in IAEA Safety Series and IAEA Technical Reports Series). This compilation can be done by integrating additional studies into geological, geochemical, biochemical and/or geophysical surveys according to the site specific conditions and requirements.

All these tasks are time and cost consuming. The burden may factually influence the competitiveness of each company since countries, states, provinces or even local communities may have different, variable strict regulations. But there is no doubt that environmental protection is not only a rhetoric or mental public issue but more so a must, in the industrialized nations and the developing nations as well.

In summary, the task and duty of an explorationist is not limited anymore to the technical search for deposits as it was in the past. In these days his obligation is extended to match the stricter regulatory requirements for conduct of work, environment protection, and to deal with the anxiety or sensitivity of the general public towards any technical operation or intervention into nature.

CURRENT DEVELOPMENTS IN URANIUM EXPLORATION ACTIVITIES IN EGYPT

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Abstract

Current developments in uranium exploration activities since 1993 are summarized, with a brief history of uranium exploration in Egypt. New targets for these exploration activities are also outlined. Previous exploration techniques used were: airborne radiometric and magnetic surveys, ground radiometric surveys, regional and detailed geologic mapping, exploratory mining works at the Wadi level, limited diamond drilling, as well as all supplementary laboratory works. Most of these activities were concentrated on the Eastern Desert terrains, particularly in granitic rocks. Main discoveries are four uranium occurrences in Pan African younger granites in addition to one at the contact of bostonite and felsite dykes in metasediments and one in psammitic gneisses in the Eastern Desert, as well as one in siltstone in a Paleozoic sedimentary basin within granitic rocks in Sinai. Two new activities are now underway: exploratory drilling programs in the uranium occurrences in the Eastern Desert and Sinai with newly acquired equipment, and experimental heap leaching of the low grade uranium ores at the site. In addition, some other techniques have been strengthened and updated such as well logging, airborne spectrometric surveys and ground geophysical surveys. Exploration activities have been recently directed also to new target areas in sedimentary formations and intracratonic sedimentary basins. The possibility of the occurrence of unconformity related deposits are also considered.

1. INTRODUCTION

Previous exploration activities in Egypt included airborne radiometric and magnetic surveys, ground radiometric surveys, regional and detailed geologic mapping, exploratory mining works at the Wadi level, limited diamond drilling, as well as all supplementary laboratory works [1]. Most of these works were concentrated on the Eastern Desert terrains, particularly in granitic rocks. Main discoveries are four uranium occurrences in Pan African younger granites in addition to one at the contacts of bostonite and felsite dykes in metasediments and one in psammitic gneisses in the Eastern Desert, as well as one siltstone in a Paleozoic sedimentary basin within granitic rocks in Sinai (Fig. 1). Exploration is still in progress to verify radiometric anomalies and test geologically favourable areas. However, none of the discovered uranium occurrences were evaluated to the stage of proven reserves. This was primarily due to lack of funding and the postponement of the nuclear program of Egypt more than once. Recently, a push was given to the Nuclear Materials Authority to enhance its activities to evaluate the discovered uranium occurrences as well as to introduce new exploration targets and production capabilities. This will also make it possible to extend the exploration activities to new areas with good potentialities in new types of uranium deposits. This brief review report will present the main points in these activities.

2. NEW ACTIVITIES

Increased budgets allocated to the Nuclear Materials Authority in the third 5-year plan (1992–1997) enabled the following exploration activities:

2.1 Airborne radiometric survey

This activity started in Egypt in the late 1950s, but was discontinued in 1973 at the end of the working periods of the survey aircrafts owned by the Nuclear Materials Authority (NMA). In the late 80's, certain areas were surveyed by the petroleum industry and the spectrometric results were

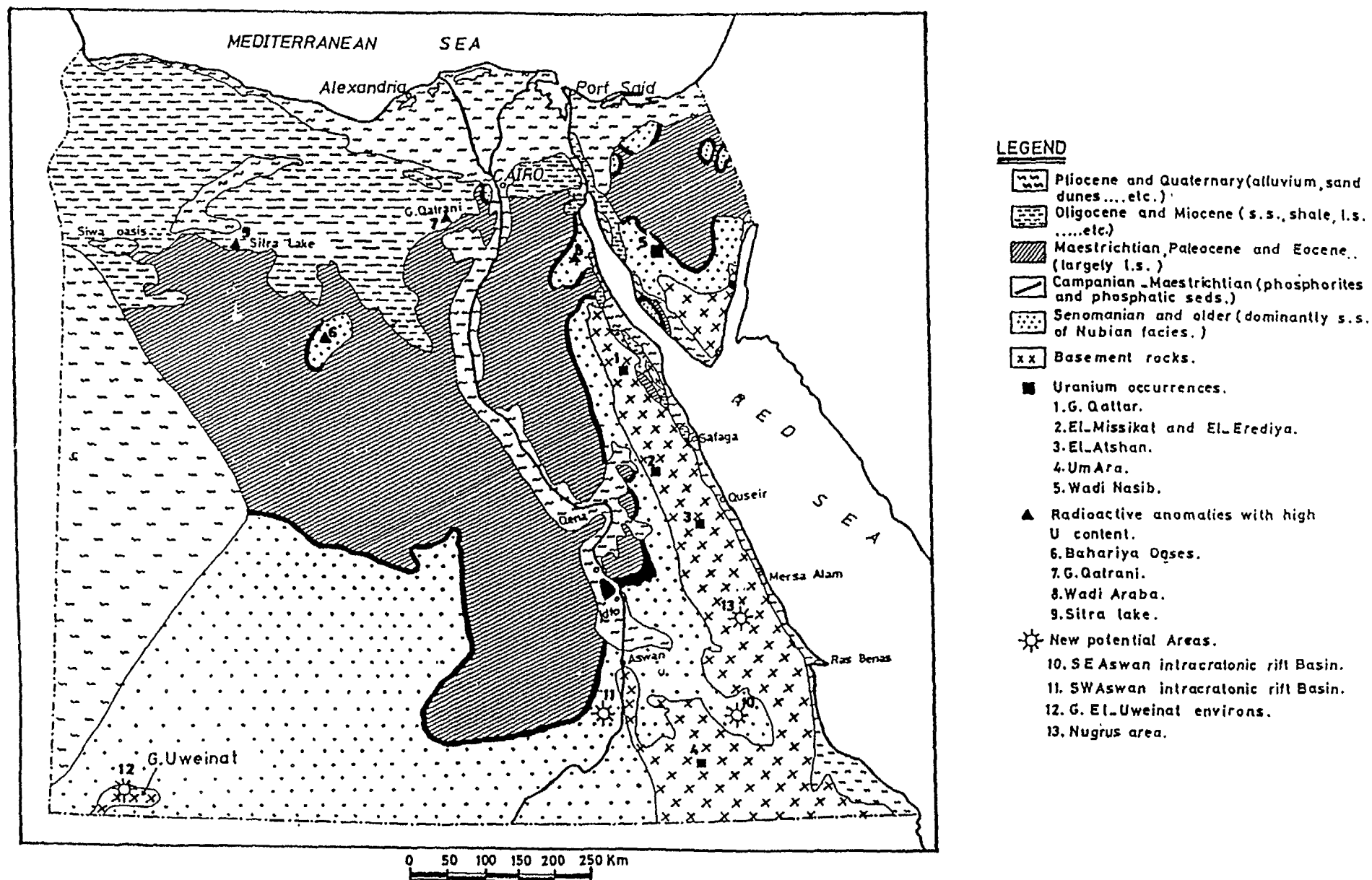


FIG. 1. A simplified geological map of Egypt with locations of uranium occurrences and areas of uranium potentialities.

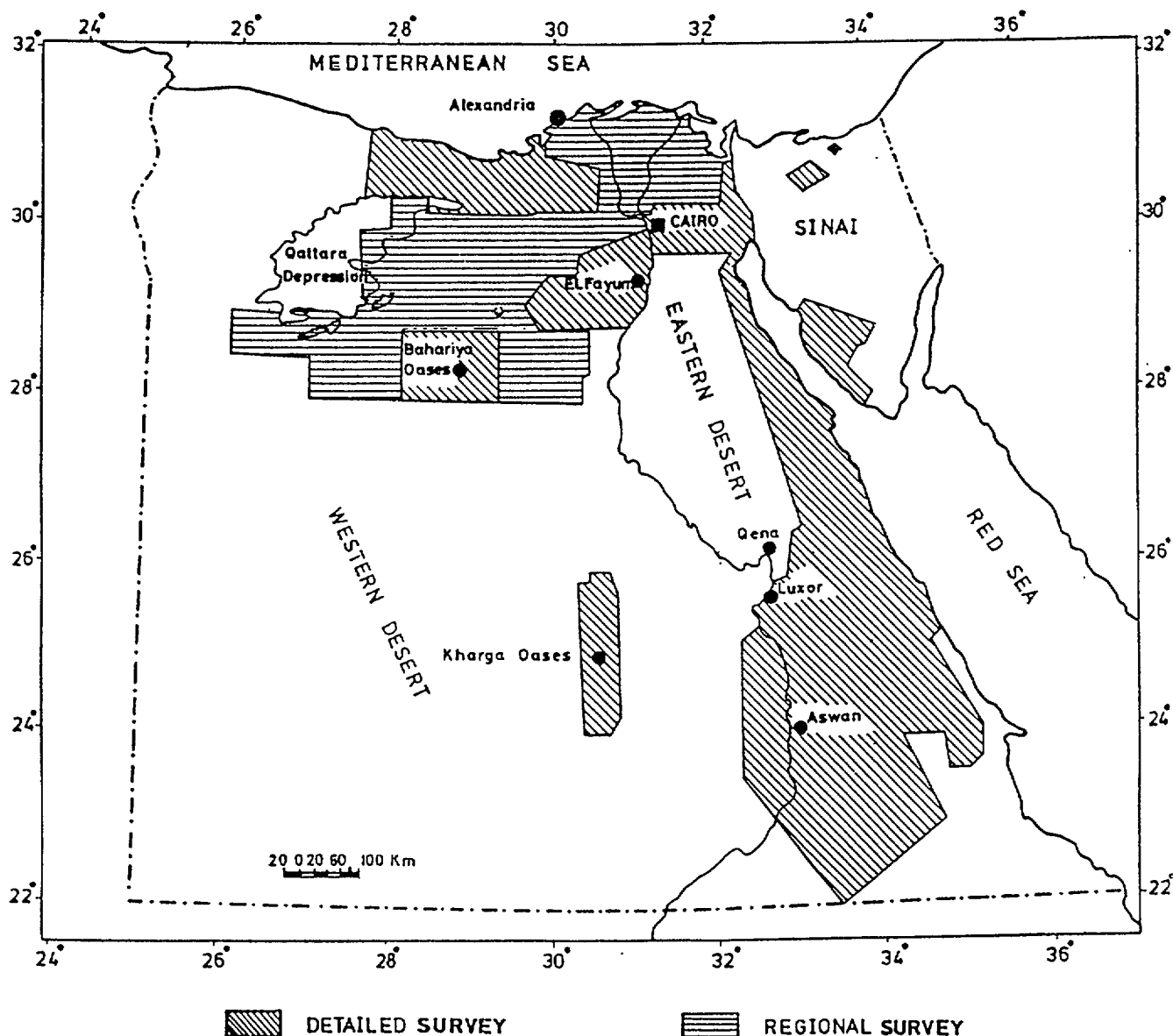


FIG. 2. A map showing the airborne radiometric and magnetic surveyed areas of Egypt.

acquired by the NMA. The total areas surveyed are shown in Fig. 2. Recently, a new aircraft with modern spectrometric and magnetic survey equipment was acquired and it is scheduled to start operations in late 1995. In the meantime, training of the survey team is underway. A 10-year plan for systematic survey of Egyptian territory is now in preparation.

2.2. Exploratory drilling programmes

Only Al Atshan uranium occurrence was drilled previously, with a total of 3600 metres diamond drill holes [2]. Exploratory mining works were carried out according to the drilling results. Lenses of uranium concentration in the form of disseminated pitchblende were located [3] (Fig. 3). Later, in the late 1970s, exploratory tunnels were excavated in Al Missikat and Al Aradiya uranium occurrences (Figs 4 and 5) to follow uraniferous shear zones in granites [4]. These tunnels intercepted sporadic lenses of primary and secondary uranium minerals. In the early 1980s, a comprehensive

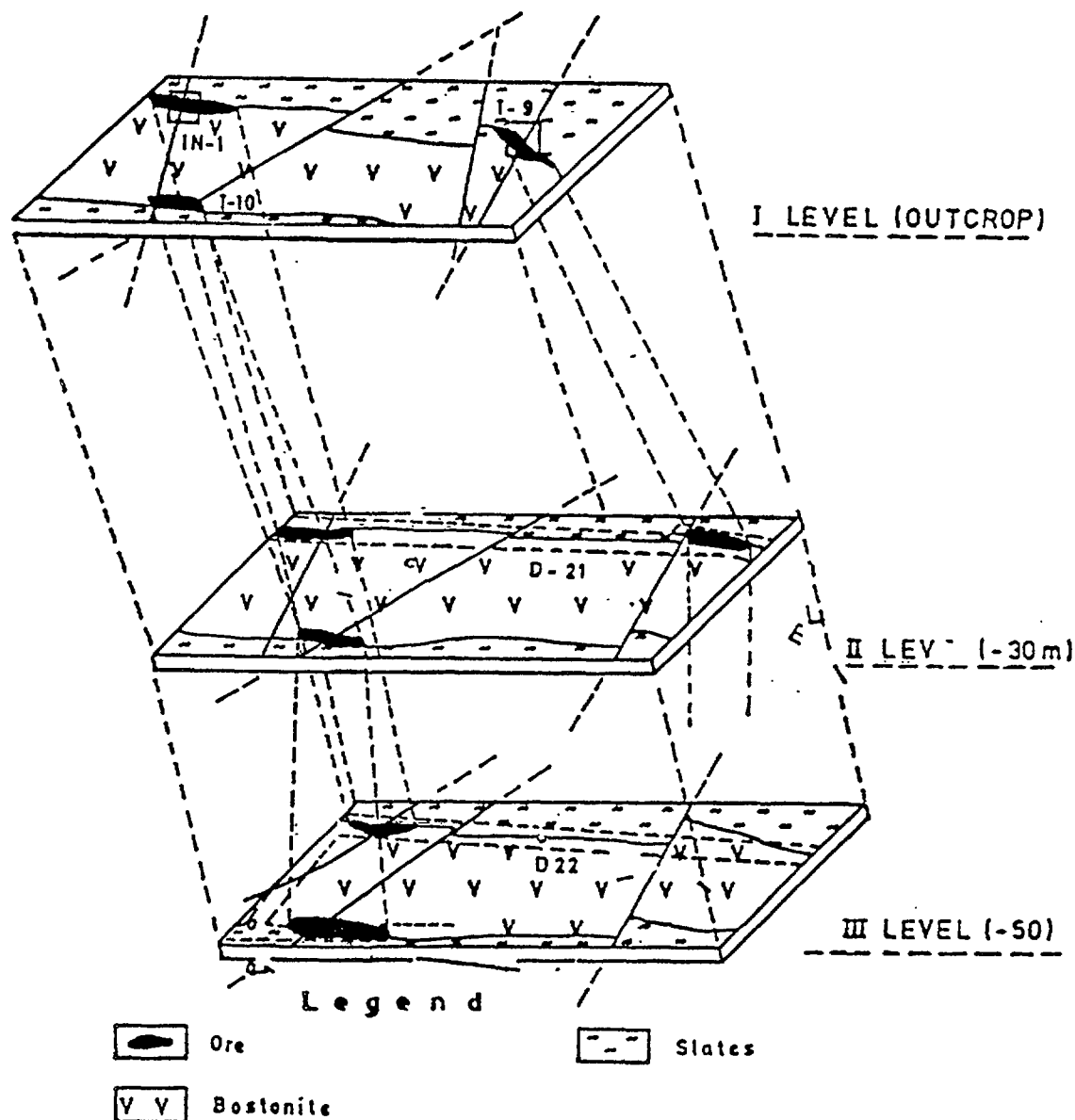


FIG. 3. Forms of the ordebodies in Wadi El Atshan locality.

percussion drilling program was negotiated with COGEMA (Figs 6 and 7) to test the occurrence of primary ore at depth, but it was not executed because of financial difficulties. Very recently, 2 new drilling rigs were acquired, and they are now in operation. The first one is drilling from the surface in Al Missikat, and the second is drilling from the underground tunnel in Al Aradiya. At the same time, we are trying to revive the percussion-drilling program which was previously negotiated with COGEMA. At the Gabal Qattar occurrence (Fig. 8) [5], several secondary uranium showings occur in a younger granite pluton at or near its contact with metasediments, which are also mineralized. Because of topographic difficulties, it was not possible to follow the uraniferous shear zone by drilling from the surface. So an exploratory drift was excavated parallel to the shear zone with cross-cuts crossing it. Several drilling stations were prepared for drilling from this drift (Fig. 9). In addition, a vertical shaft is proposed at the northern extension of that shear zone where high concentration of secondary mineralizations were recorded. From this vertical shaft, two cross-cuts will be driven parallel to the shear zone to explore the occurrence of primary mineralization at depth. Several newly discovered lenses of secondary mineralization at the contact of Qattar pluton with the Hammamat metasediments raises the potentiality of this area.

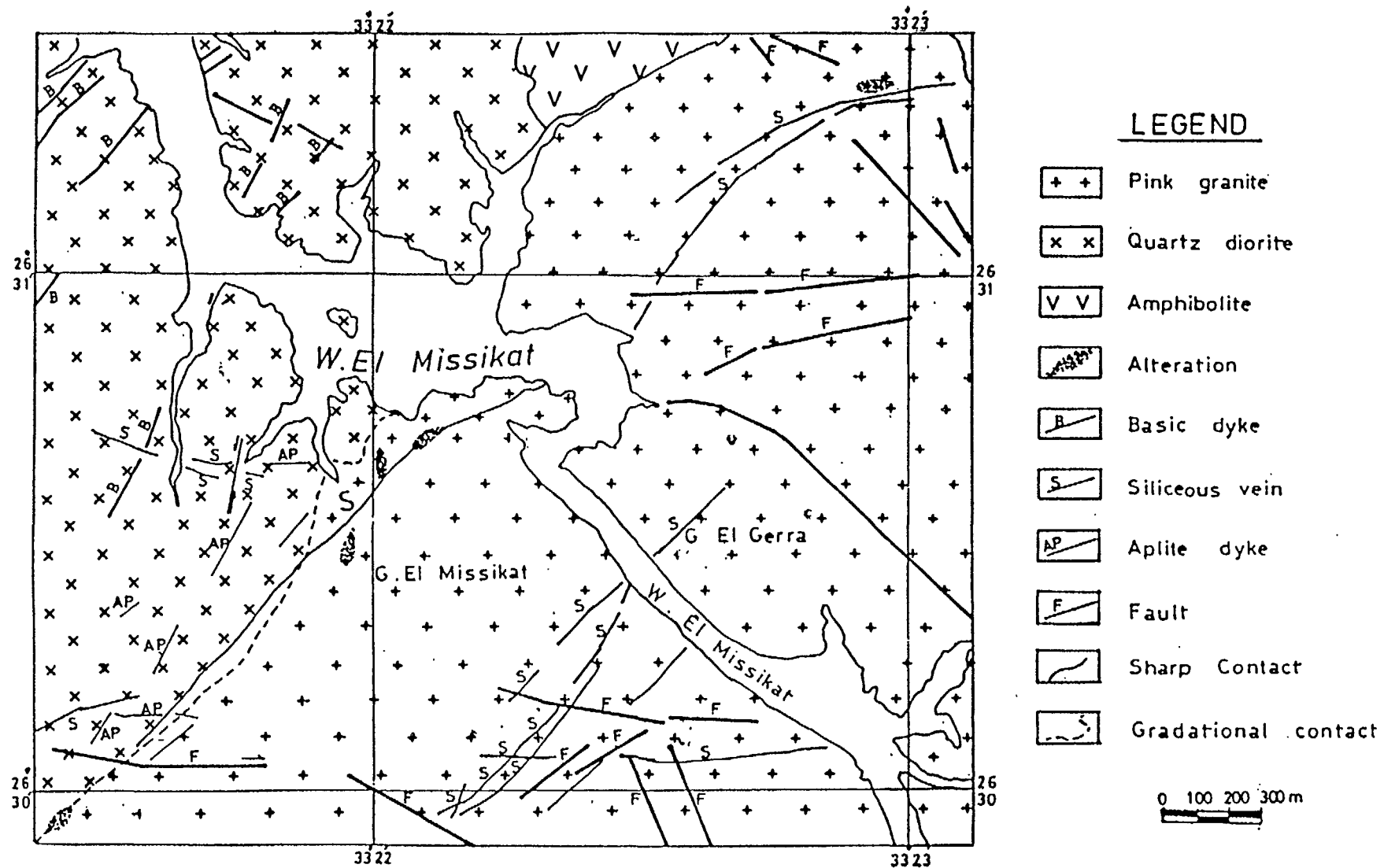


FIG. 4. Geological map of the northern part of Al Missikat granite showing the main siliceous vein.

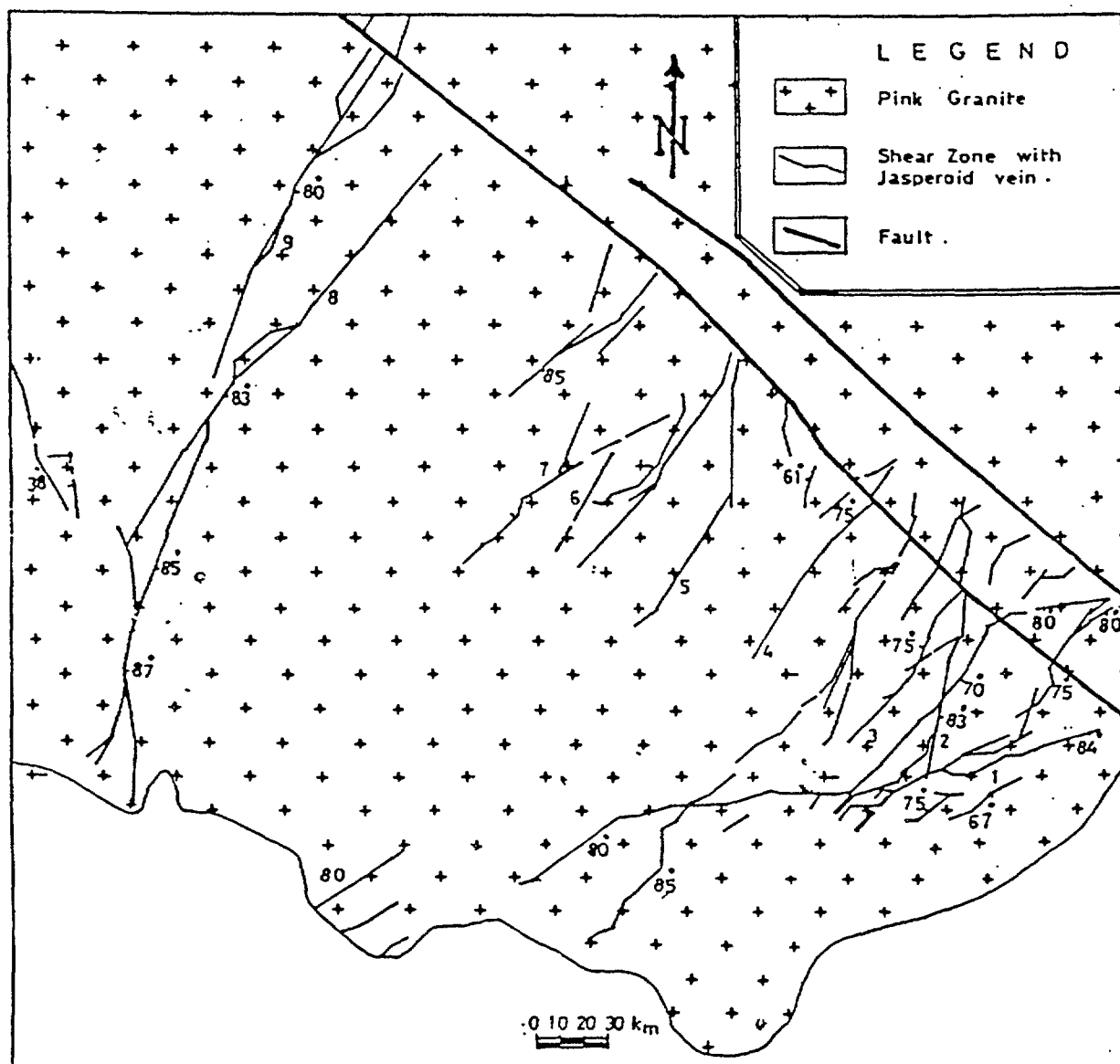


FIG. 5. The main shear zones in Al Aradiya granite.

2.3 Experimental heap leaching of low grade ores

Since in situ heap leaching of low grade uranium ores has increased (11% of uranium production last year in Europe was by heap leaching), we thought to gain the experience of this process and add it to the activities of NMA. With some consultations and expert advice from IAEA, heap leaching experiments were proposed for the excavated uraniferous granite from the exploratory trenches at Um Ara occurrence. Work is actively in preparation now. Another area proposed for heap leaching is Abu Zienima in west central Sinai [6] where secondary mineralization occurs in siltstones.

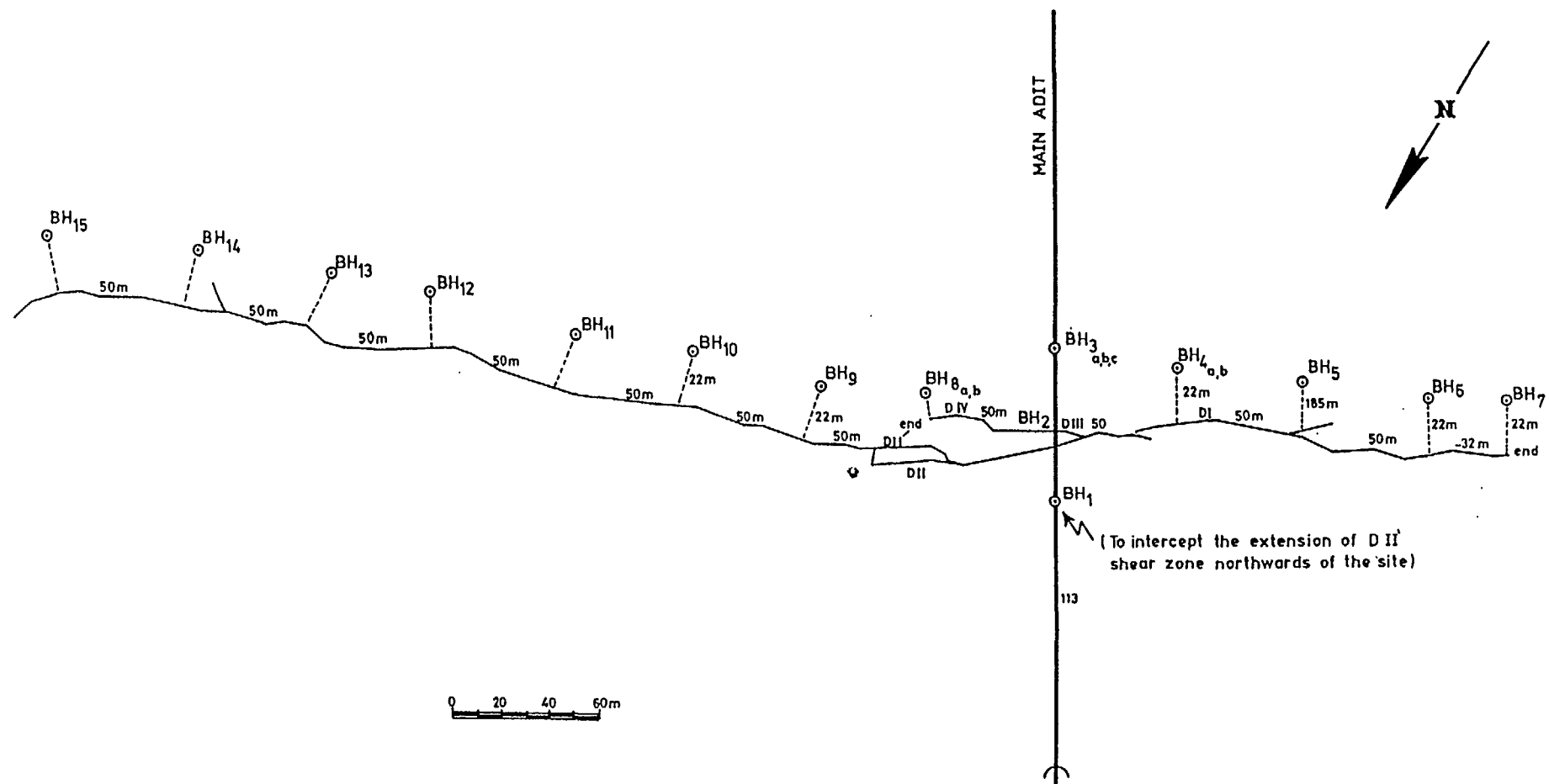


FIG 6. Plan of percussion drilling from the tunnel of Al Missikat.

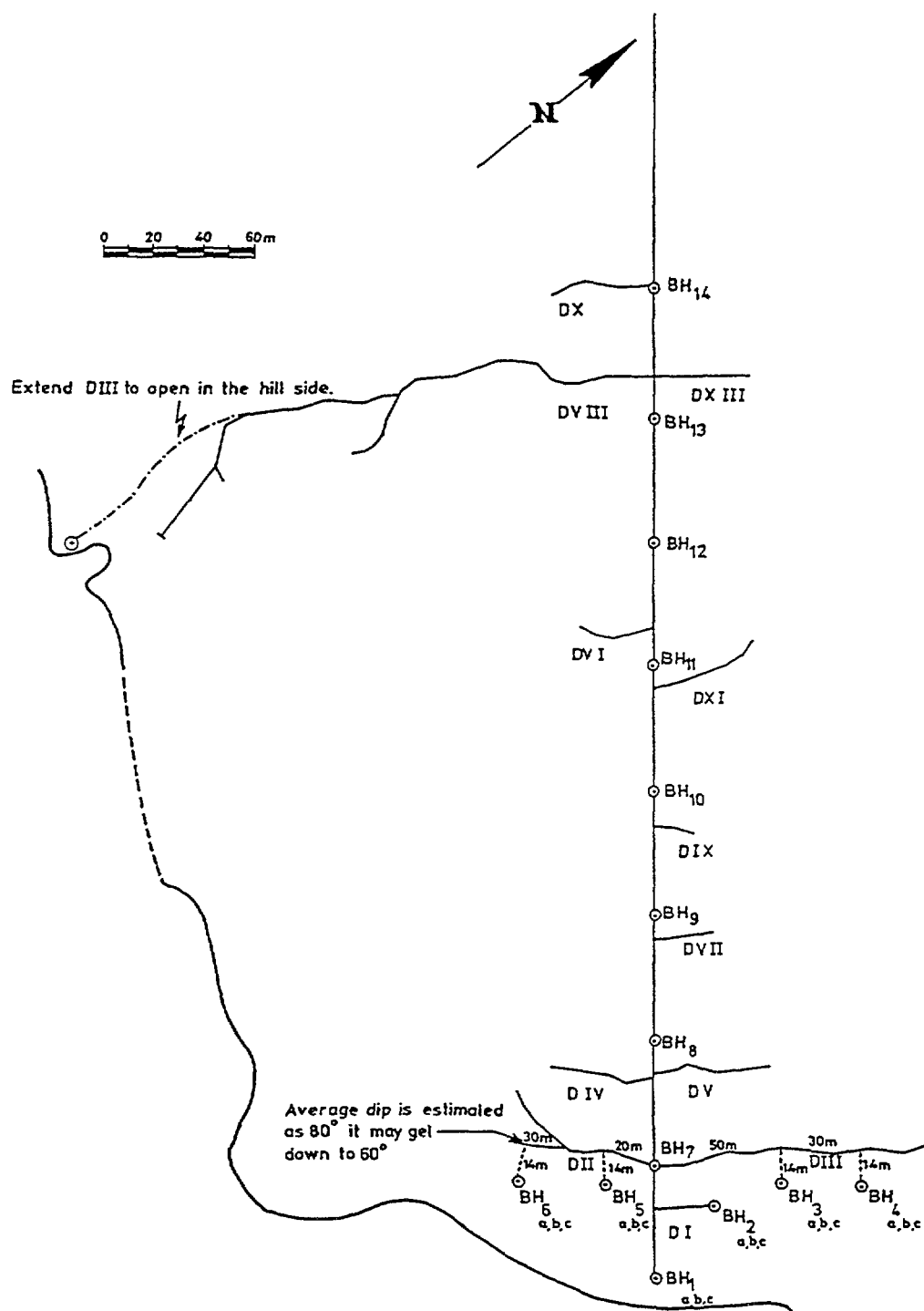
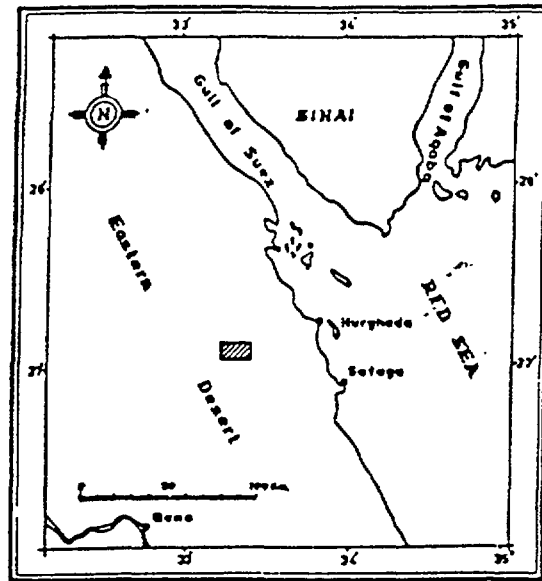


FIG. 7. Plan of percussion drilling from the tunnels of El Erediya.



KEY MAP SHOWING GEBEL QATTAR URANIUM PROSPECT

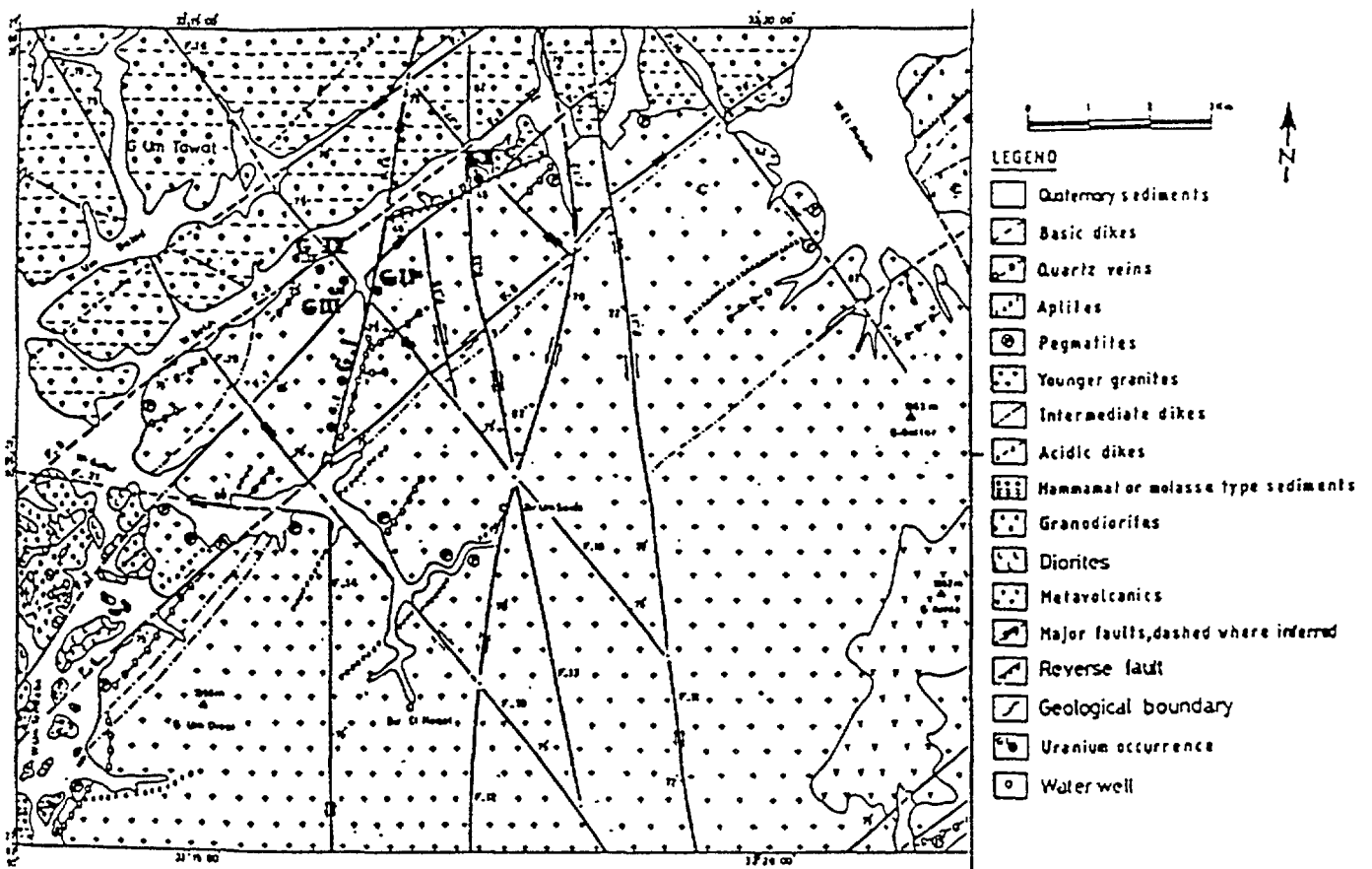


FIG. 8. Geologic map showing uranium occurrences in Gebel Qattar prospect, north Eastern Desert.

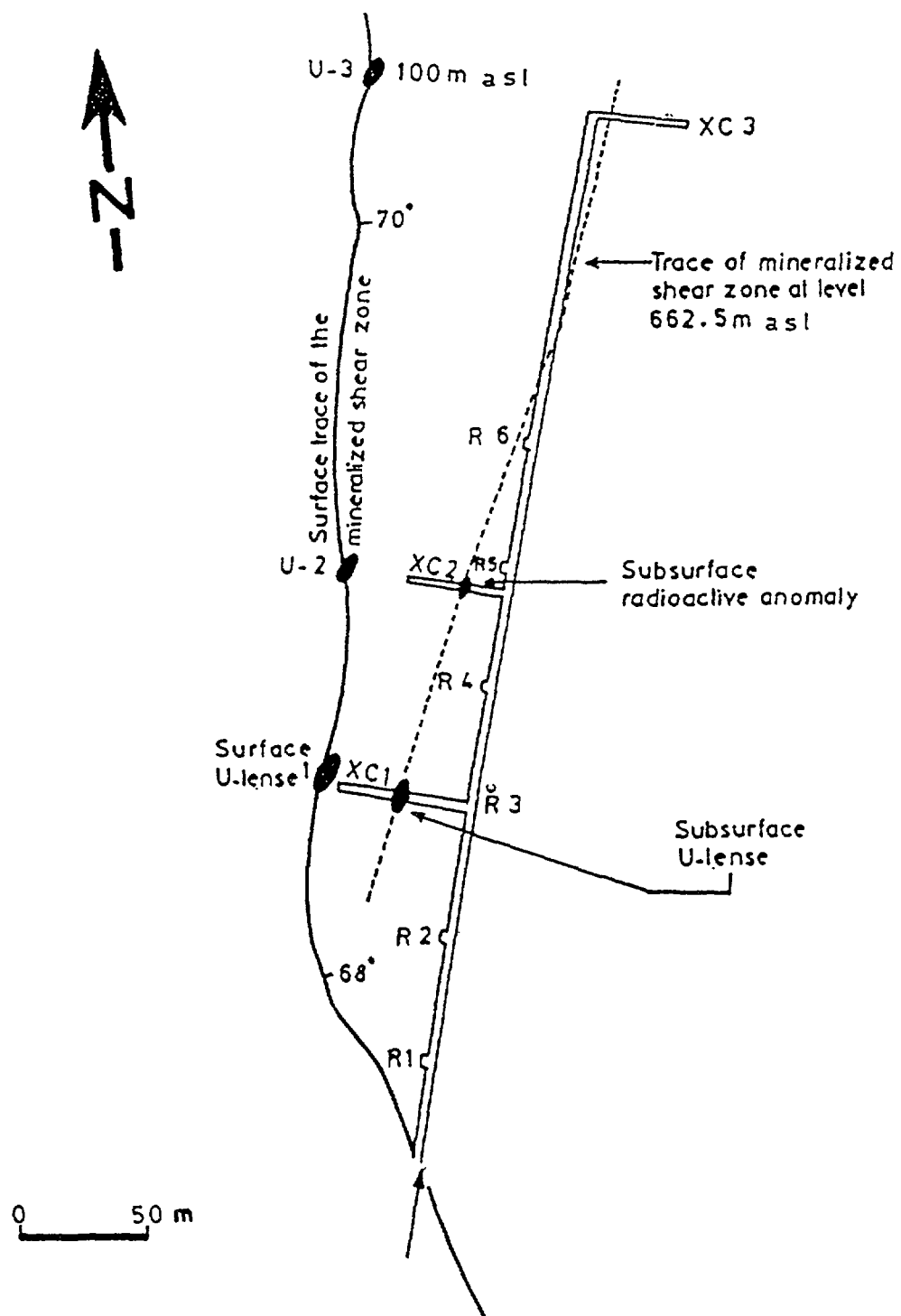


FIG. 9. Map of the exploration mining project in Gebel Qattar. Occurrence showing the uraniferous lenses.

3. NEW TARGETS FOR EXPLORATION

Previous exploration activities were specially concentrated on the granitic rocks in the Eastern Desert. Recently, other potential zones have been considered. Two of these potential zones are:

- (1) Sandstone hosted uranium deposits:
Two extensive clastic successions characterize the stratigraphic column in Egypt. The lower one is Senonian and older in age and is dominated by fluviatile sandstone formations which contain plant remains. In Wadi Araba area (location 8 in Fig. 1), a carboniferous succession dominated by sandstone occurs along a structural depression between two high plateaux to the north and south. The outcrops of the clastic beds are highly oxidized. Several high radiometric anomalies associated with high uranium contents were recorded in these beds [7]. Exploratory drilling is proposed in this site. The upper clastic succession of Eocene to Miocene age contains a large proportion of sandstone formations, which are considered a potential target particularly above oil and gas fields in the north Western Desert.
- (2) Unconformity related uranium deposits:
Two potential environments of these deposits are considered [8]. The first is the unconformity between the basement and the sedimentary cover which belong to the lower clastic succession in the Eastern Desert, south Sinai and the southeastern extremity of the Western Desert (Fig. 1). The second one occurs within the basement itself between upper Proterozoic and questionable middle Proterozoic rocks.

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COMPARATIVE ANALYSIS OF URANIUM RESOURCES OF WOCA AND NON-WOCA COUNTRIES

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Abstract

This paper presents a comparative analysis of uranium resources of WOCA and NON-WOCA countries, where the information of the database of the Russian Ministry of Atomic Energy is used. Resources are defined as Reasonably Assured Resources (RAR) and Estimated Additional Resources (EAR). Uranium deposits are subdivided into eight types according to their geological settings. In NON-WOCA countries very few deposits related to Precambrian were found although the potential for these deposits exists. They represent a large resource in WOCA countries. The deposits related to Paleozoic represent 11 % of the world resources in the NON-WOCA countries. The largest number of deposits in the NON-WOCA are classified as vein type deposit while the largest resources are located in deposits of the sandstone type and metasomatite type. The analysis of all information suggests that the most prospective areas for the discovery of large deposits are related to large synclises, taphrogenic basins, areas of tectonic-magmatic activation of ancient platforms.

Data concerning uranium resources distribution according to deposits type, geological structure and ore formation epochs have always been used for forecasting and exploration of uranium deposits. Not once these data were mentioned in the IAEA reports, but all these reports considered WOCA countries. Only recently some information about uranium deposits in the territory of the former USSR and socialist countries (Non-WOCA) has appeared (1). In this report we present the results of a detailed analysis of uranium resources distribution, where the information of the database of the Russian Ministry of Atomic Energy is used. Analyses of WOCA countries' uranium deposits were made to the end of 1992. Similar methods were used for the analyses of the NON-WOCA countries' uranium deposits. It gives an opportunity to carry out comparative analysis of resources distribution of two country groups and for the entire world. Only worked out and prospected resources were taken into consideration. According to the IAEA definition these resources are of group — A-Known Conventional Resources. It includes the following resources:

1. Reasonably Assured Resources (RAR)
2. Estimated Additional resources (EAR)

From the last group, the first category resources (EAR-1) were taken into consideration. According to Russian classification these resources were similar to those of categories C, C and P. The cost of production 1 kg of uranium concentrate includes the reserves of the group with the cost of less than 130 \$.

The workable types of uranium deposits are identified according to the IAEA classification but with some necessary changes and additions.

1. Sandstone deposits: stratiform deposits, connected with oxidation fronts or with ground oxidation processes, localized in permeable rocks of platform cover.
2. Quartz-Pebble conglomerate deposits: deposits similar to those of Witwatersrand (South Africa) and Elliot-Lake (Canada).

3. Unconformity-Related deposits: deposits similar to those of the Athabaska region (Canada) and Pine-Creek (Australia).
4. Vein type deposits. These deposits include classical veins (vein filled of opened cavities) and veined-stockwork deposits. The last one is often accompanied by zones of metasomatic and dissemination of uranium minerals.
5. Unique deposits represented by Olympic-Dam in Australia and deposits of Ronnenburg ore field in Germany.
6. Disseminated metasomic type deposits: deposits, where the ore is represented by newly formed metasomatic minerals and the concentration of relict minerals of initial rocks is less than 10%. The largest deposits of this type are known to be located on the Ukrainian and Aldan (Russia) shields.
7. Granitic and pegmatitic deposits: the Rossing deposit (Namibia) and Bancroft (Canada) and their analogues.
8. Deposits of other type. Here we can include mineralized breccia pipes, karsts, uranium-contained black clays, deposits of uranium-containing phosphatized bone detritus of fossil fish, stratiform deposits in ancient sedimentary rocks in the Francville (Gabon) and Onezhsky (Russia) basins.

All in all there are 508 uranium deposits registered in the territories of Non-WOCA countries (USSR, GDR, Czechoslovakia, Bulgaria, Hungary, Romania, Poland, Mongolia). Table I shows deposits distribution according to the resources amount.

TABLE I. DISTRIBUTION OF DEPOSITS ACCORDING TO RESOURCE AMOUNT

Uranium reserves, tonnes	Number of deposits	% of reserves
Up to 1 000	233	1.94
1 000–10 000	188	16.01
10 000–50 000	71	35.95
50 000–100 000	11	17.09
More than 100 000	5	29.01
TOTAL	508	100.00

17.95% of uranium resources are concentrated in 421 small deposits (up to 10 000 t), 35.95% — in 71 average deposits (10 000–50 000 t) and the concentration of uranium resources in 16 large and very large deposits is 46.1%. We can associate 29% of this country group (Non-WOCA) uranium resources with three deposits of sandstone type and two deposits of disseminated metasomatite type.

Table II shows the distribution of uranium resources according to types of deposits.

The largest number of deposits are represented by Vein type. Among them veined-stockwork deposits are prevailing (249 deposits), which is equal to 21.7% of uranium resources. The major part of resources of the classical vein deposits is located in the Shlema-Alberoda deposit (Germany) and Prshibram (Czech Republic). The concentration of 50 other deposits is equal to only 1.7% of uranium resources.

TABLE II. DISTRIBUTION OF URANIUM RESOURCES IN % ACCORDING TO TYPE OF DEPOSIT (NON-WOCA COUNTRIES)

Type of deposit	Number of deposits	% of resources
1. Sandstone type	112	40.8
2. Quartz-Pebble conglomerates	1	0.04
3. Unconformity-related	-	-
4. Vein and similar to vein	301	26.5
5. Unique	25	4.2
6. Disseminated metasomatic	41	24.8
7. Disseminated granitic and pegmatic	3	0.4
8. Others	25	3.3
TOTAL	508	100.00

Quartz-Pebble conglomerates (type 2) are represented by only one small deposit (Nickolo-Kozelskoye) in the Krivorozhsky basin in the Ukraine. In the Pobuzhsky region (Ukraine) three small deposits in pegmatites have been prospected. However, there are several hundreds of ore occurrences of the quartz pebble type in Non-WOCA countries that have not been fully prospected.

The Unique deposits (type 5) are represented by 15 deposits of the Ronnenburg ore field (Germany), outside which only few small deposits of similar type were found.

Among other types of deposits (type 8) one can pay special attention to the uranium deposits on the Mangyshlack Peninsula (Kazakhstan), which are associated with phosphatized bone detritus of fossil fish in paleogenic sediments.

One can carry out comparative analysis of uranium resources distribution both in the two country groups and in the world according to the information given in Table III. It is absolutely obvious that for all above mentioned country groups sandstone type deposits (type 1) are the most significant. The major part of uranium resources is connected with them and they can be exploited by comparatively cheap method of in situ leaching.

Large sums of money were spent for searching quartz-pebble conglomerate deposits, but it was practically in vain. And there is practically no chance in the future to find new deposits of this type.

Large and rich uranium deposits of unconformity-related type (type 2) have been recently discovered in Canada and Australia, and perspectives of finding similar type of deposits are very realistic. Much attention is paid to this problem in Russia and the Ukraine.

The well-known vein-type deposits (type 4) in the territories of WOCA countries have been practically worked out. But one should face the problem of searching for widely spread in Russia, Kazakhstan and the Czech Republic veined-stockwork deposits (type 6), which contain considerable uranium resources in the territories of Non-WOCA countries. The same is true regarding metasomatic deposits, widely spread the Ukraine (disseminated albitites) and Aldan (disseminated potash metasomatites) shields. These deposits are rather large and usually their resources exceed 25 000 tonnes. The content of uranium in ores is ordinary.

TABLE III. URANIUM RESOURCES DISTRIBUTION (IN %) ACCORDING TO THE TYPES OF DEPOSITS IN WOCA AND NON-WOCA COUNTRIES

Types of deposits	WOCA	Non-WOCA	Worldwide
1. Sandstone	42.7	40.8	41.7
2. Quartz-pebble conglomerates	17.3	0.04	8.4
3. Unconformity-related	15.4	-	7.5
4. Vein type:			
– classical veins	3.4	4.8	4.1
– streaky-stockwork	5.4	21.7	13.8
5. Unique	7.9	4.23	6.0
6. Metasomatic	3.10	24.8	14.3
7. Disseminated granitic pegmatic	2.8	0.4	1.5
8. Other types	2.0	3.3	2.7

The presence of large unique (from the geological stand point) deposits (type 5) is evidence of possibility of finding their analogues and possibility of discovery of some new, unknown types of uranium deposits.

Distribution of uranium resources according to the ore formation epochs is shown in Table IV. Uranium ore in most deposits have been formed and redeposited throughout several epochs. In the table they are attributed to the very epoch, when the major part of the given deposit ores were formed. For example, uranium minerals in conglomerates are dated 3100–1000 mln.y. Conglomerates of the Krivorozhsky basin were formed during Transvaal-Guron epoch (2500 ± 100 mln.y.), with which we associate the formation of the major part of ores in Witwatersrand and Elliot-Lake deposits. Later, all these ores were subjected to redeposition within Belomorian and Grenvillian epochs, and as a result the major part of them reached radiological age of $2\ 000 \pm 100$ mln.y.

All world uranium deposits have been formed practically simultaneously within 13 epochs of ore formation. These epochs represent rather short periods of time. It is very difficult to identify their duration precisely because of the errors in determining radiological age of uranium ores. The most part of uranium world resources is associated with Precambrian (31%) and Mesozoic (34%) epochs of ore formation; 11% is associated with Paleozoic epoch and 24% with Cenozoic epoch.

In Non-WOCA countries in comparison with WOCA ones, very few Precambrian deposits were found. The resources of these deposits are 5 and 26% of the world amount, respectively. Absolutely different picture is observed in the deposits of Paleozoic age. Such deposits are not known in the territories of WOCA countries and their concentration in Non-WOCA countries is equal to 11% of world resources.

The most endogenic deposit type is associated with ancient epochs of ore formation. Exogenic deposits of sandstone type were formed only in younger epochs, and only vein deposits are practically found in all epochs beginning with Early Proterozoic and up to Cenozoic epochs.

Interesting data were obtained regarding the distribution of uranium resources in different geological structures (Table V). More than 70% of world uranium resources are associated with the deposits, localized in depression structures, 40% of which are concentrated in the deposits of platform cover, represented mainly by sandstone type, and 30% in the places of taphrogenic basins. The largest deposits of sandstone type are concentrated in the synclise of platform cover (37.4% of world

TABLE IV. URANIUM RESOURCES DISTRIBUTION (IN %) DIFFERENT DEPOSIT TYPES AMONG DIFFERENT EPOCHS OF ORE FORMATION (1 - WOCA, 2 - NON-WOCA COUNTRIES, 3 - IN THE WORLD)

Epochs of ore foration	Age, mln.y	Types of deposits (table 3)									
		1	2	3	4	5	6	7	8	The whole	
1. Belomorian	2000+	1	8,41					-	0.43	8.83	
	100	2	-					0.19	-	0.19	
		3	8,41					0.19	0.43	9.03	
2. Hudsonian	1770+	1			0,71	0.11			-	0.82	
	50	2			-	4,83			0.48	5.31	
		3			0,71	4,94			0.48	6.13	
3. Elsonian	1500+	1			0.36	3,84	0.14			4.34	
	100	2			-	-	-			-	
		3			0.36	3,84	0.14			4.34	
4. Grenvillian	1000+	1		7.46	0.48			0.16		8.10	
	100	2		-	-			-		-	
		3		7.46	0.48			0.16		8.10	
5. Katangian	650+	1			0.43					0.43	
	50	2			-					-	
		3			0.43					0.43	
6. Damarian	500+	1			0.85		1.28	1.17		3.30	
	50	2			-		-	-		-	
		3			0.85		1.28	1.17		3.30	
7. Caledonian	390+	1			-					-	
	20	2			2.79					2.79	
		3			2.79					2.79	
8. Early Varissian	310+	1			-					-	
	40	2			2.23					2.23	
		3			2.23					2.23	
9. Late Varissian	270+	1			0.43	-			0.27	0.70	
	30	2			3.21	2.17			-	5.38	
		3			3.64	2.17			0.27	6.08	
10. Cimmerian	180+	1	14,24		0.26		-		0.28	14,78	
	40	2	0.82		5.27		7.93		-	14.02	
		3	15.06		5.53		7.93		0.28	23.80	
11. Early Alpine	70+10	1	-		0.63				-	0.63	
		2	2.85		0.11				1.21	4.17	
		3	2.85		0.74				1.21	4.80	
12. Late Alpine	20+5	1	4.93		0.12					5.05	
		2	17.17		-					17.17	
		3	22.10		0.12					22.22	
13. Resent	0-5	1	1.61						-	1.61	
		2	0.11						0.03	0.14	
		3	1.72						0.03	1.75	

TABLE V. URANIUM RESOURCES DISTRIBUTION (IN %) OF DIFFERENT DEPOSIT TYPES IN DIFFERENT GEOTECHTONIC ELEMENTS OF THE EARTH'S CRUST (1 - WOCA, 2 - NON-WOCA, 3 - IN THE WORLD)

No	Geotectonic elements of the crust			Types of deposits (table 3)										
				1	2	3	4	5	6	7	8	The whole		
1	Depressions		Syneclises	1 18,3							0,4	18,7		
					2 19,1							1,2	20,3	
					3 37,4							1,6	39,0	
2		Platform cover	Intermount basins	1 0,9									0,9	
					2 1,1								1,1	
					3 2,0								2,0	
3			Paleovalleys	1 1,7									1,7	
					2 0,8								0,8	
					3 2,5								2,5	
4			Taphrogenic basins	1	8,50	7,6	0,7	3,9				0,4	21,1	
					2	0,02	-	4,9	2,2				0,5	7,6
					3	8,5	7,6	5,6	6,1				0,9	28,7
5	Structures of the basement	Fold belts	1			0,9		0,1	1,3			2,4		
				2			1,8		-	0,2		1,9		
				3			2,7		0,1	1,5		4,3		
6		Middle massifs	1			0,9						0,9		
				2			6,0					6,0		
				3			6,9					6,9		
7		Areas and Zones of tectono-magmatic activation	1			2,0		1,5		0,2		3,5		
				2			-		12,9		-	12,8		
				3			2,0		14,4		9,2	16,3		

resources) and some deposits of less significance in intermount basins (2%) and in paleovalleys (2.5%).

The largest number of different deposits types (5 from the total of 8) are concentrated in taphrogenic basins. Vein type deposits are not so "strict" about geological structures. They are concentrated in all structures, except for platform cover.

Disseminated granitic and pegmatitic deposits are associated with the completing stage of foldbelts zone development, and metasomatic deposits are associated with the areas and zones of tectono-magmatic activation.

Analysis of all given above information gives the opportunity to make conclusion regarding the most perspective geotectonic elements for discovering large uranium deposits.

Generally, the most prospective areas are:

1. Large synclises of platform cover, constructed by permeable mesocenozoic sediments, containing stratiform deposits of sandstone type.

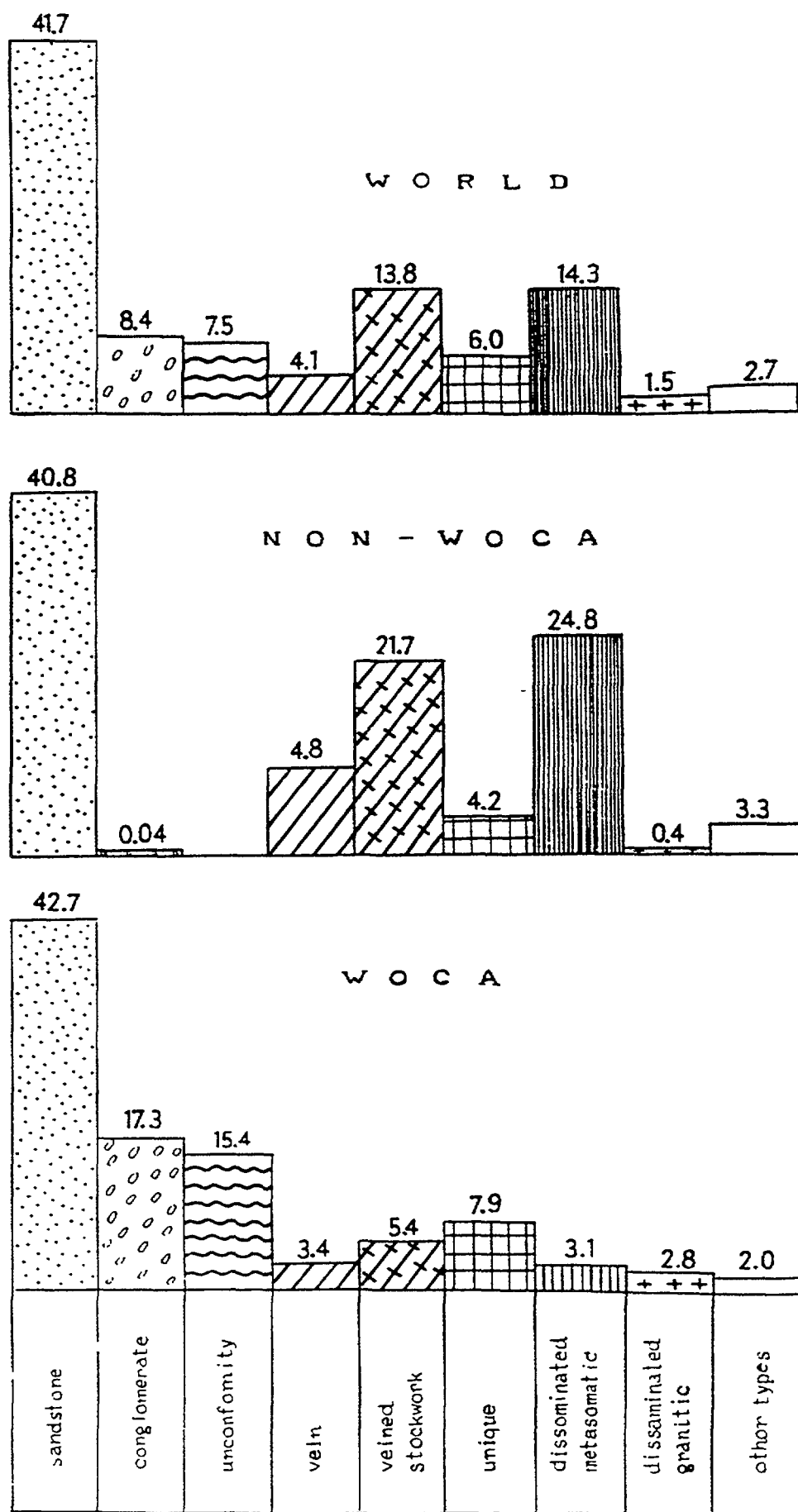


FIG. 1. Uranium resources of various deposit types (in %).

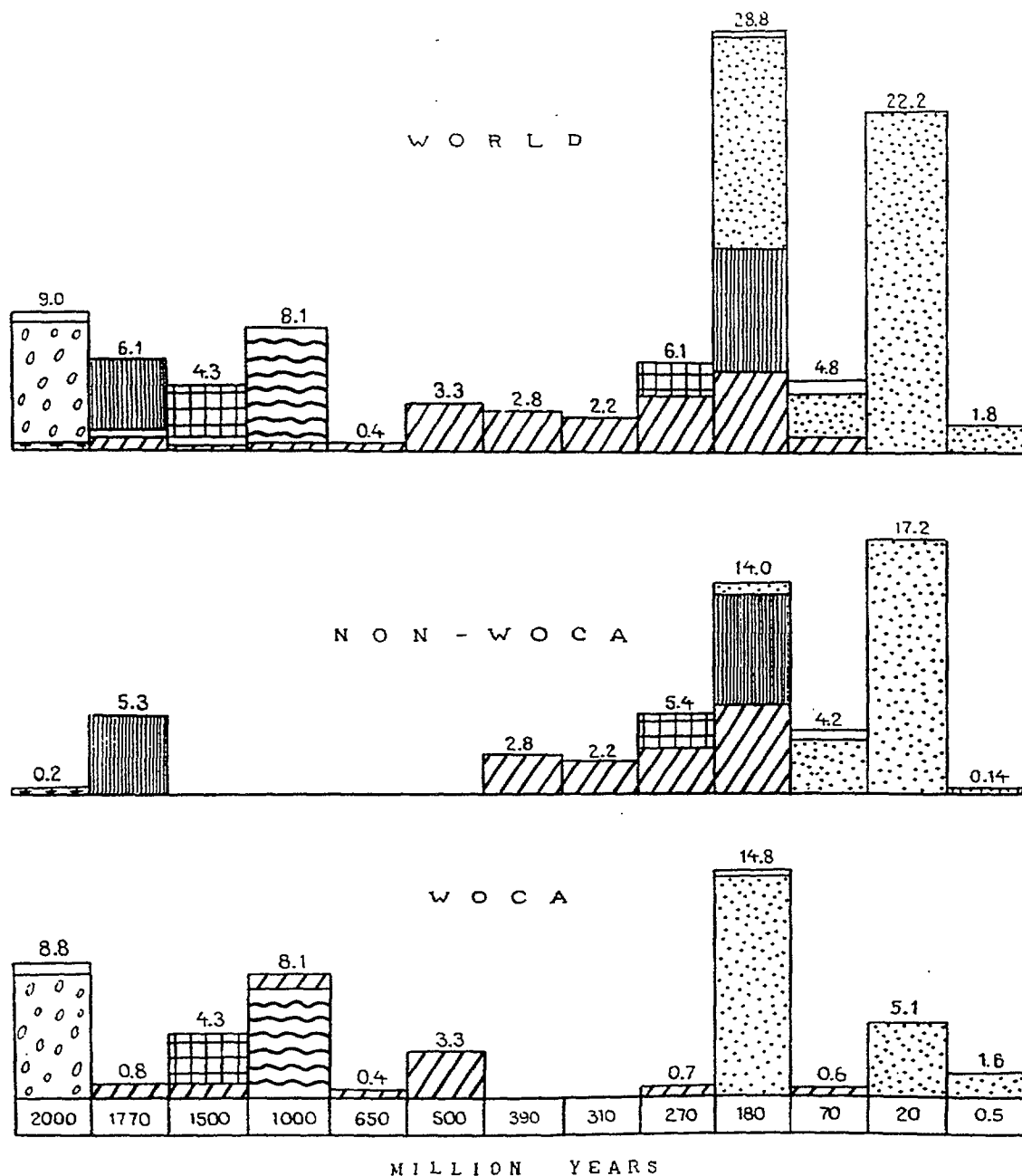


FIG. 2. Uranium resources of various ore formation epochs and deposit types (in % from the world resources); for the legend see Fig. 1.

2. Taphrogenic basins in early Proterozoic basement, constructed by late Proterozoic volcano-sedimentary rocks with unconformity-related deposits.
3. Taphrogenic basins with a different age ratio of basement and overlapping sediments, where large uranium deposits of new workable types could be found, including those similar to the Olympic-Dam and Ronnenburg ore field deposits.
4. Middle massifs inside the Caledonic and Variscian foldbelts with large deposits of vein and veined-stockwork types.
5. Areas and zones of Tectono-magmatic activation of the ancient platforms shields with large metasomatic deposits of Middle Proterozoic and Mesozoic age.

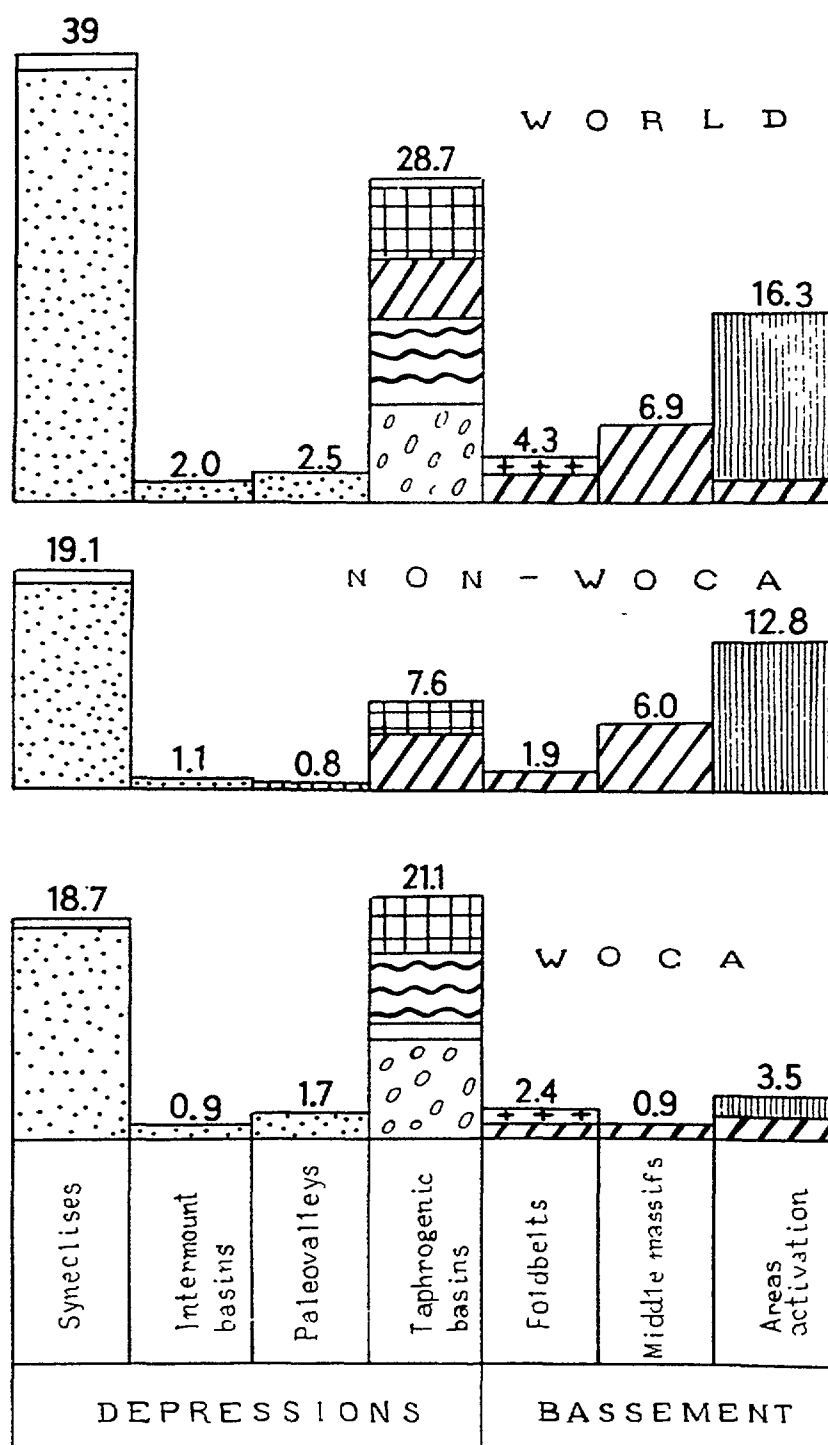


FIG. 3. Uranium resources of various geological structures and deposit types (in % from the world resources).

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ON THE POSSIBILITY OF IDENTIFYING LOW COST, MEDIUM GRADE URANIUM DEPOSITS CLOSE TO THE PROTEROZOIC UNCONFORMITY IN THE CUDDAPAH BASIN, ANDHRA PRADESH, INDIA

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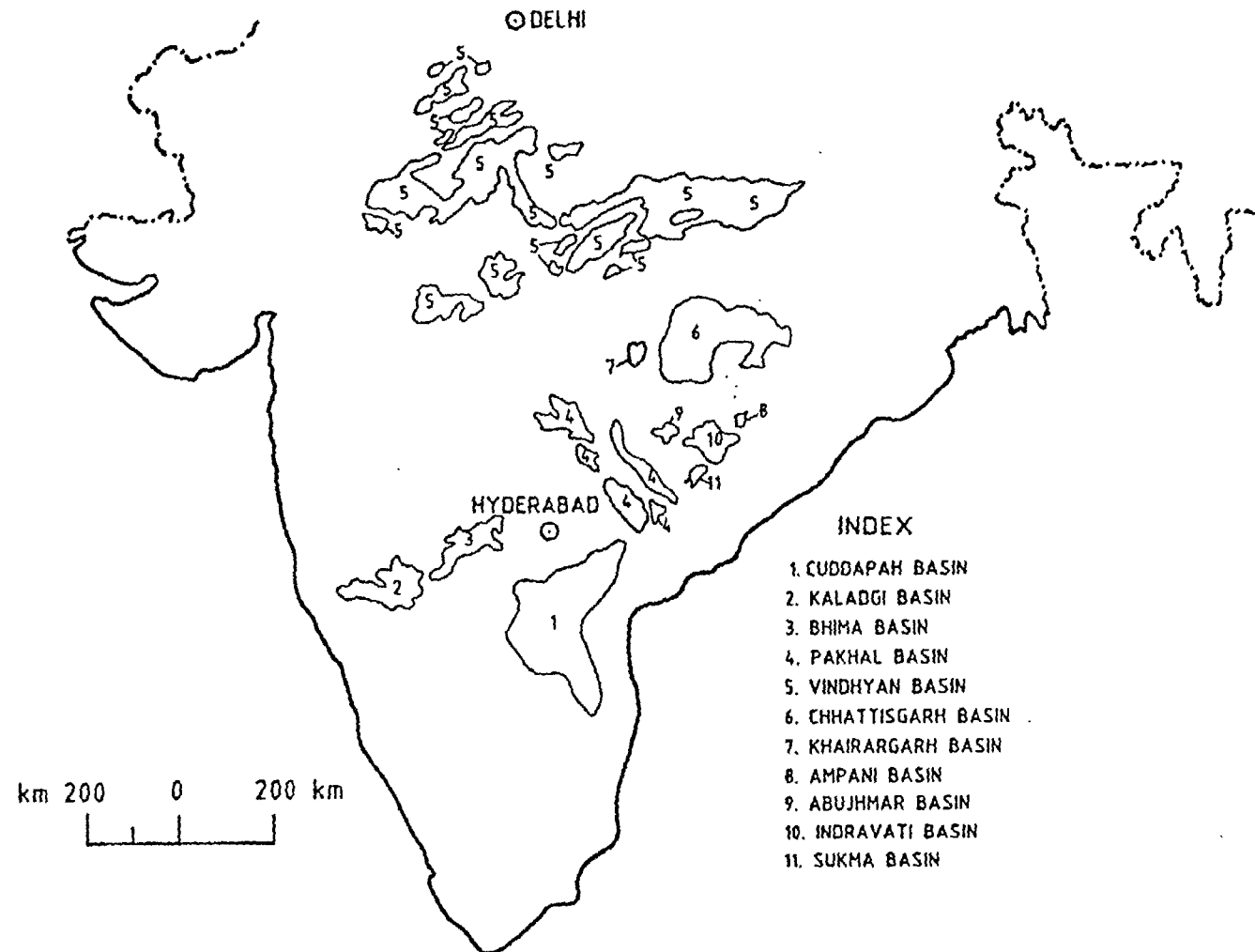
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Abstract

With increasing emphasis on unconformity-type uranium deposits within intracratonic Middle-Proterozoic basins, as documented in Canada and Australia, efforts towards identifying such basins in the Indian Peninsular Shield, since 1990, have encountered some success. The Mid-Proterozoic Cuddapah Basin has thus come up as a most promising target area in India. On the north-western margin of this basin the Srisaillam Formation, the youngest member of the Cuddapah Supergroup, directly overlies the basement granite of Lower Proterozoic/Archaean age and forms a dominant plateau of more than 3000 km². Amongst the numerous uraniferous anomalies located close to this unconformity the Lambapur occurrence has been under active exploration. Uranium mineralization at Lambapur is essentially confined to the basement granite and occurs in the form of elongate pods at the intersections of two prominent sets of fractures, trending NNE-SSW and NW-SE, with the unconformity plane. The ore bodies have very sharp outlines with very shallow depth persistence (5 m) below the unconformity. Three sets of basic dykes intrude the basement granite in NNE-SSW, E-W and NW-SE trends. The NNE- and E-W trending dykes are also mineralized wherever sheared and fractured. Association of uraninite with druzy quartz, galena, chalcopyrite and pyrite point to possible hydrothermal nature for this mineralization. The Lambapur occurrence, although of low to medium grade (0.060 to 0.30% U₃O₈), is unique in the sense that it has many of the features akin to the classical unconformity-type deposits of Canada and Australia. Many of the important parameters such as the age of the unconformity and mineralization, age and role of basic dyke swarms, and associated alteration patterns are under study. On the margins of the Cuddapah basin, uranium mineralization has also been identified to be associated with numerous fractures/faults within the granite. Some of them are partly explored and show shallow depth persistence and low-to medium grades — a feature well established at Lambapur. These fracture/fault-controlled mineralization probably represent relicts of Lambapur-type mineralization, now exposed owing to removal of cover rocks. The Lambapur discovery has enabled closer scrutiny of the numerous uraniferous anomalies and the geological set-up of the Middle Proterozoic Cuddapah Basin and established this intracratonic basin as a new exploration target area for unconformity-type deposits in India.

1. INTRODUCTION

The high-grade and large-tonnage unconformity-related uranium deposits of Canada and Australia are now well understood and documented [5, 6, 15, 16]. These vein-like uranium deposits occur as fracture/breccia-fillings in the Lower Proterozoic pelitic rocks (\pm graphite) close to the unconformity, under the cover of dominantly arenaceous Middle Proterozoic strata. With a view to locate similar deposits in India, renewed ground radiometric surveys were initiated in 1990 on the northwestern margin of the Middle Proterozoic Cuddapah Basin. This led to the discovery of numerous uraniferous anomalies, close to the unconformity between the Archaean/Lower Proterozoic Mahboobnagar granite and Srisaillam Formation, in a number of dissected outliers. The most promising anomaly at Lambapur, Nalgonda district, Andhra Pradesh, is under detailed investigations since 1992. This discovery represents a significant break-through and may contribute substantially to the uranium resource of India. More significantly, the geological understanding of this unique mineralization may give definite clues in locating the classical unconformity-type deposits in the northern parts of the Cuddapah Basin.



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 "PURANA BASINS OF PENINSULAR INDIA"

FIG. 1. Middle to upper proterozoic basins of peninsular India.

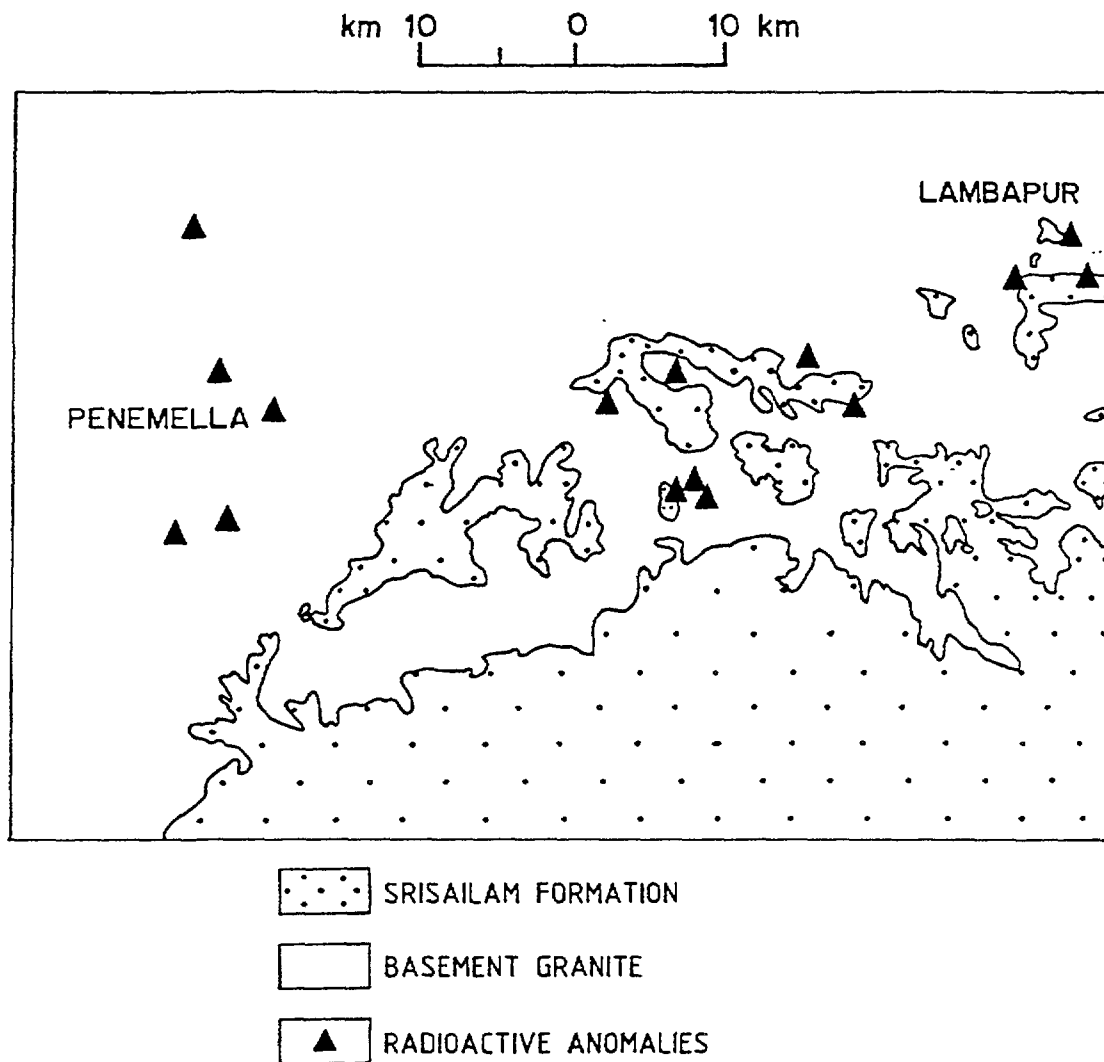


FIG. 3. Geological map of part of northern margin of Cuddapah Basin, Nalgonda and Mahbubnagar District, Andhra Pradesh.

2. GEOLOGY AND STRUCTURE

2.1. Regional geological setting

The crescent-shaped Cuddapah Basin (Figs 1 and 2), having an areal extent of 44 000 km², contains over 12 km thick sequence of sedimentary and volcanic rocks belonging to the Middle-Upper Proterozoic Cuddapah Supergroup and Kurnool Group. The western margin of the basin is marked by a non-conformity with the formations resting on the Archaean gneisses. The eastern margin shows a thrust contact where the older Archaean gneisses/Dharwar metasedimentary rocks are thrust over the rocks of the Cuddapah basin. The Cuddapah Supergroup is predominantly arenaceous to argillaceous with subordinate calcareous to dolomitic units, whereas the Kurnool Group mainly consists of carbonate facies sediments [4]. The stratigraphic succession is given in Table I.

TABLE I. LITHOSTRATIGRAPHY OF CUDDAPAH SUPERGOURP AND KURNOOL GROUP

SUPERGROUP	GROUP	FORMATION	DOMINANT LITHOLOGY
C U D D A P A H S U P E R G R O U P	KURNOOL GROUP		CARBONATE FACIES WITH SUBORDINATE FINE CLASTIC
	----- U N C O N F O R M I T Y -----		
		SRISILAM QUARTZITE	QUARTZITE AND SHALE
	----- U N C O N F O R M I T Y -----		
	NALLAMALAI	CUMBUM FORMATION	PHYLLITE/SLATE QUARTZITE, DOLOMITE
		BAIRENKONDA QUARTZITE	QUARTZITE, SJALE
	-- A N G U L A R U N C O N F O R M I T Y --		
	CHITRAVATI GROUP	GANDIKOTA QUARTZITE	QUARTZITE, SHALE
		TADPATRI FORMATION	SHALE, TUFF, QUARTZITE, DOLOMITE WITH INTRUSIVES
		PULIVENDLA QUARTZITE	CONGLOMERATE AND QUARTZITE
	----- D I S C O N F O R M I T Y -----		
	PAPAGHNI GROUP	VEMPALLE FORMATION	STROMATOLITIC DOLOMITE, CHERT BRECCIA WITH BASIC FLOWS/INTRUSIVES
		GULCHERU QUARTZITE	CONGLOMERATE QUARTZITE, SHALE
	----- N O N - C O N F O R M I T Y -----		
ARCHEAN GNEISSES AND DHARWAR SCHISTS			

(After Nagaraja Rao, et al. 1987)

2.2. Srisailam Sub-basin

Srisailam Formation, the youngest member of the Cuddapah Supergroup, forms a very prominent plateau, having an areal extent of over 3000 km², in the north-western part of the Cuddapah Basin and has been designated as the Srisailam sub-basin. The Srisailam Formation is mainly an arenaceous unit alternating with shale, with sub-horizontal dips, and attains a maximum thickness of 300 m. It directly overlies, on its northwestern margin, the basement rocks comprising Archaean gneisses and younger granites (Mahboobnagar granite) with ages ranging from 2268 ± 32 Ma to 2482 ± 70 Ma [7]. On the southern and eastern margins, the Srisailam Formation covers the Nallamalai Group with an angular unconformity.

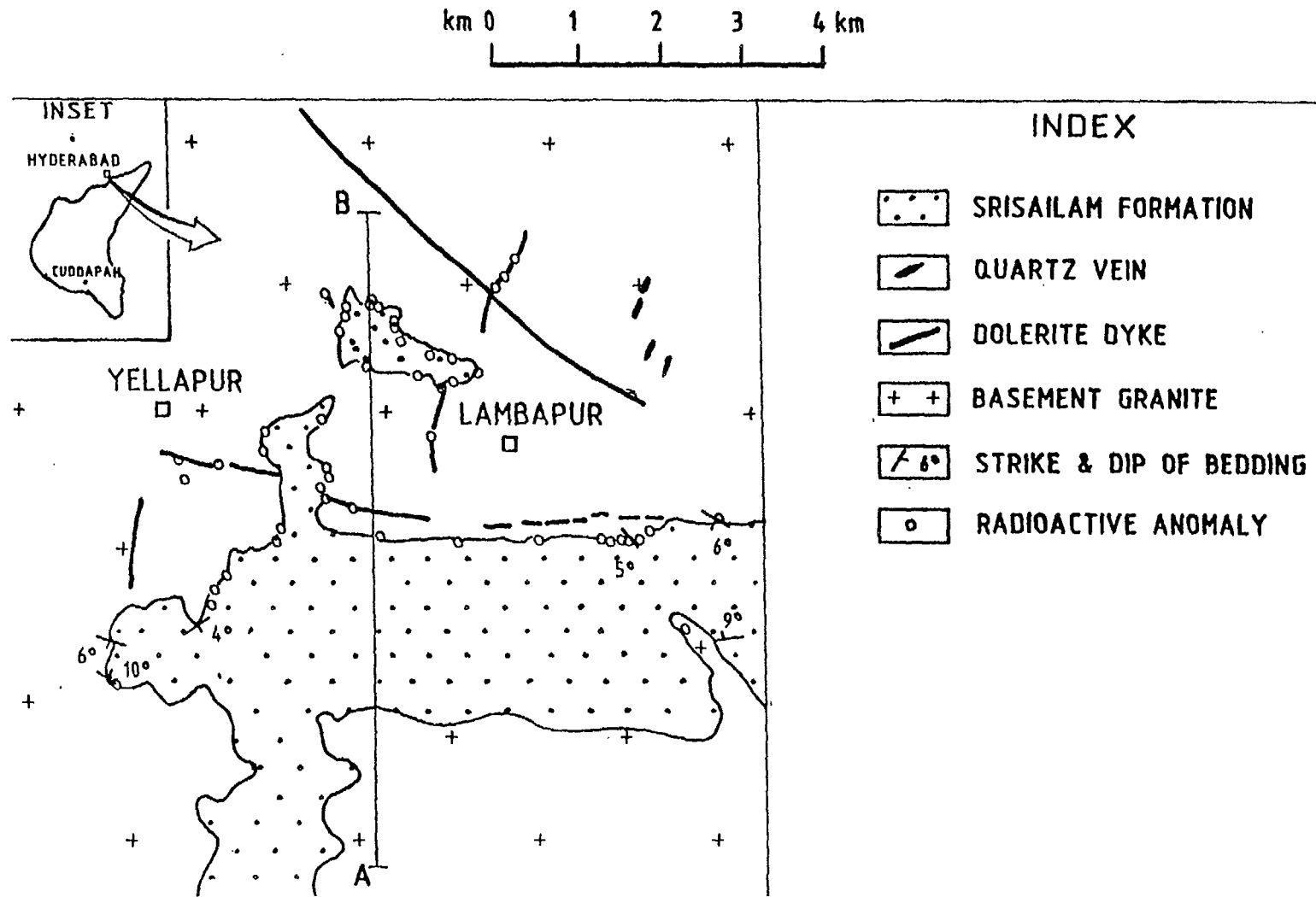


FIG. 4. Geological map of Lambapur area, Nalgonda District, Andhra Pradesh.

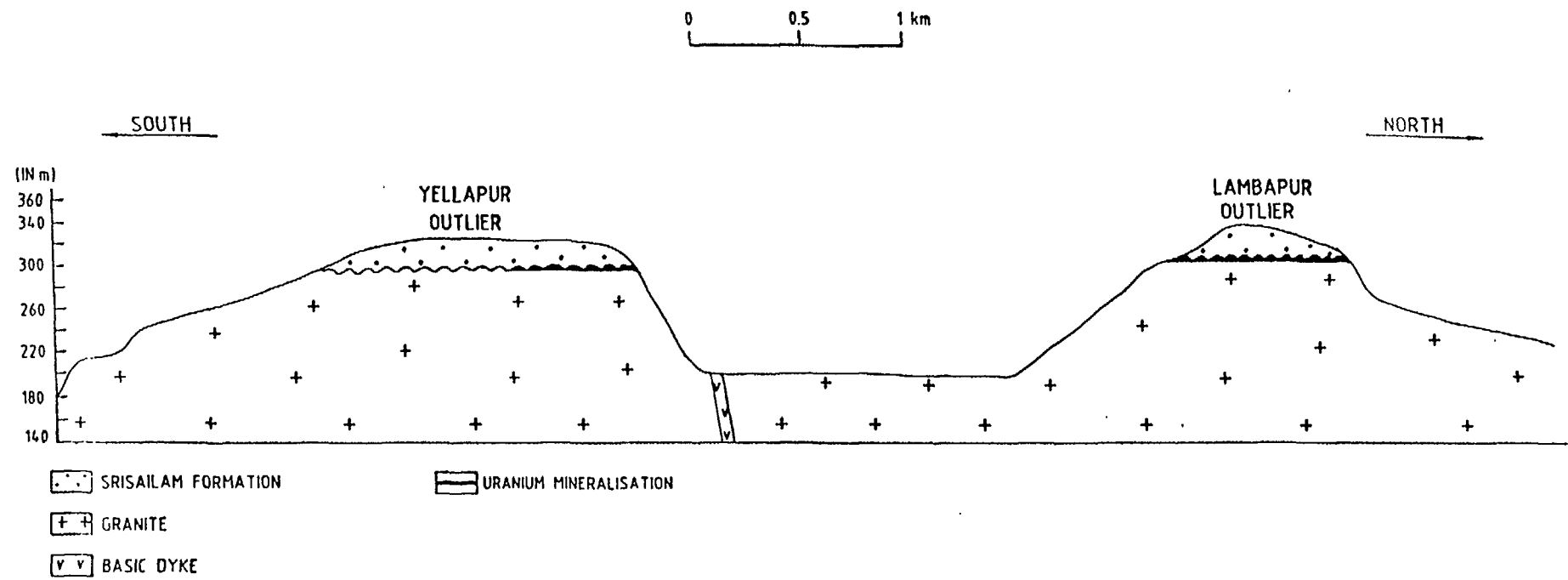


FIG. 5. Profile section across Lambapur and Yellapur outliers.

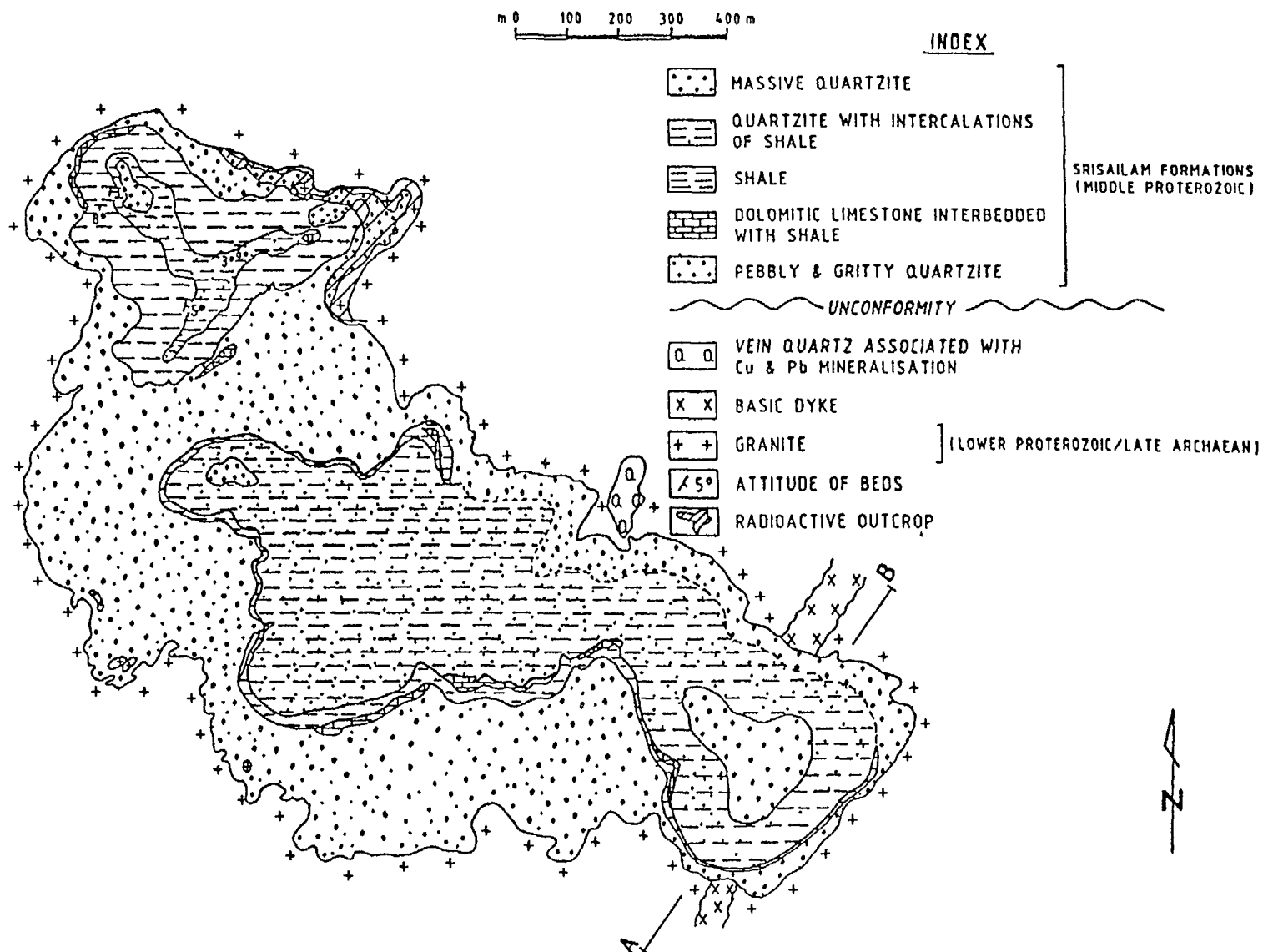


FIG 6 Geological map of Lambapur Ridge, Nalgonda District, Andhra Pradesh

2.3. Geological setting of Srisailam outlier at Lambapur

The Srisailam sub-basin has a highly dissected topography on its north-western fringes, resulting into the development of numerous isolated flat-topped hills rising up to 100 to 150 m above the ground level. Lambapur outlier is one such flat-topped hill, having a dimension of 1.8 km by 0.70 km, and exposes the unconformity between the Mahboobnagar granite and Srisailam Formation. In this outlier the thickness of the Srisailam Formation varies from 1 m to 50 m and is represented by the lithological units as shown in Table II. These lithological units show a gentle dip of 3° to 5° towards southeast. The basement granite is traversed by three sets of basic dykes, trending NNW-SSW, NW-SE and E-W as well as by NNE-SSW trending quartz vein with associated sulphides (Figs 4 and 5).

TABLE II. THE SRISAILAM FORMATION AND ITS LITHOLOGICAL UNITS

MID PROTEROZOIC	CUDDAPAH SRISAILAM SUPERGROUP FORMATION	VEIN QUARTZ WITH PB AND CU MINERALIZATION
		MASSIVE QUARTZITE
		QUARTZITE WITH SHALE INTERCALATIONS
		SHALE WITH LIMESTONE INTERCALATIONS
		PEGGLY/GRITTY ARENITE
UNCONFORMITY		
L. PROTEROZOIC/ ARCHAEAN		GRANITE/GNEISSES WITH INTRUSIVE YOUNGER DYKES

At Lambapur the extent of the main uraniferous outcrop is only 300 m on the north-western fringes and only isolated mineralized outcrops are observed on the remaining parts of the outlier. These anomalies are associated with both the basement granite and the basal member of the Srisailam Formation close to the unconformity (Fig. 6).

3. LAMBAPUR URANIUM MINERALIZATION

3.1. Exploration history

Prior to 1990 the emphasis was given to identify anomalies associated with granites and look for possible vein-type mineralization associated with them. Thus, ground radiometric surveys, since the late 1950s, resulted in discovering numerous uraniferous anomalies associated with the Mahboobnagar granite and intrusive basic dykes within them, on the north-western fringes of the Cuddapah Basin [1, 10, 11, 12].

With the advent of unconformity-type uranium deposit, renewed efforts were initiated in the form of ground radiometric surveys since 1990 to look for similar set-up available in the Cuddapah Basin. This led to the discovery of numerous uraniferous anomalies close to the unconformity between the Archaean/Lower Proterozoic Mahboobnagar granite and the Srisailam Formation in the Lambapur and adjoining outliers [8, 9, 14, 17] (Fig. 3). These anomalies were thus distinctly different from those discovered in the earlier phases of surveys. The most promising outcrop anomaly at Lambapur (Nalgonda district, Andhra Pradesh) is under detailed exploration since March, 1992. The adjoining Yellapur outliers discovered subsequently, has also indicated the presence of numerous uraniferous anomalies at the unconformity interface (Fig. 4).

The outcrop samples of the Srisailam Quartzite, granite and dolerite dyke from mineralized outcrops, close to the unconformity, from the Lambapur-Yellapur area assayed significant uranium values with negligible thorium. Some of the results are given in Table III. These samples show disequilibrium both in favour of parent uranium and daughter products.

TABLE III. SOME OUTCROP SAMPLE RESULTS OF LAMPAPUR AREA

Sample No.	Rock type	Radiometric assay data		
		%eU ₃ O ₈	%U ₃ O ₈	%ThO ₂
R-8	Granite	0.08	0.03	0.01
R-16	Granite	0.02	0.01	<0.005
R-18	Granite	0.3	0.49	0.01
R-42	Granite	0.24	0.44	<0.005
R-9	Quartzite	0.042	0.046	<0.005
R-27	Quartzite	0.34	0.5	<0.005
R-43	Quartzite	0.021	0.025	<0.005
R-33	Basic dyke	0.089	0.089	<0.005
R-34	Basic dyke	0.024	0.025	<0.005
R-35	Basic dyke	0.37	0.45	<0.005

3.2. Exploratory drilling

In view of the lack of adequate exposure due to the cover of Srisailam Formation and its limited thickness (1 m to 55 m), it was thought ideal to explore the Lambapur outlier by shallow percussion drill holes for a quick appraisal (Fig. 7). For this purpose the entire outlier of 1.8 km × 0.70 km was covered on square (100 m × 100 m) to rectangular (150 m × 100 m) grid pattern. Infilling core-drilling, at selected locations, has been corroborated the results of non-core drilling.

On the basis of the results of the drilling the sub-surface correlation, the nature and controls of uranium mineralization and ore-reserves were established and are discussed.

3.3. Nature of ore body and controls of uranium mineralization

At Lambapur the uranium mineralization is found close to the unconformity, both in the basement granite and the overlying Srisailam pebbly arenite (Figs 6 and 7). Basic dykes and vein quartz within the basement granite are also mineralized close to the unconformity. The vein quartz, in addition, has associated lead and copper mineralization. The main uranium ore body is confined to the unconformity between the basement granite and the overlying Srisailam Formation. In addition, there are isolated additional mineralized zones much deeper within the basement granite. The major part (more than 85%) of the main ore body occurs in granite, close to the unconformity, at shallow depth (at 1 to 55 m from the surface) and shows a gentle dip of 3° to 5° towards south-east. This slope is more or less consistent with the basement slopes. Initially this ore body was perceived to be a blanket along the unconformity, but analyses and interpretation of drilling data indicate that the richer trends (ore-shoots) are confined to definite trends, which are NNE-SSW and NW-SE (Fig. 8). In the southeastern part of the Lambapur outlier, the main ore-body is picked up again after a barren gap. Here also the richer trends (ore-shoots) have a NNE-SSW trend and occurs parallel to the NNE-trending basic dyke, which is highly sheared on its eastern margins with intense quartz-veining.

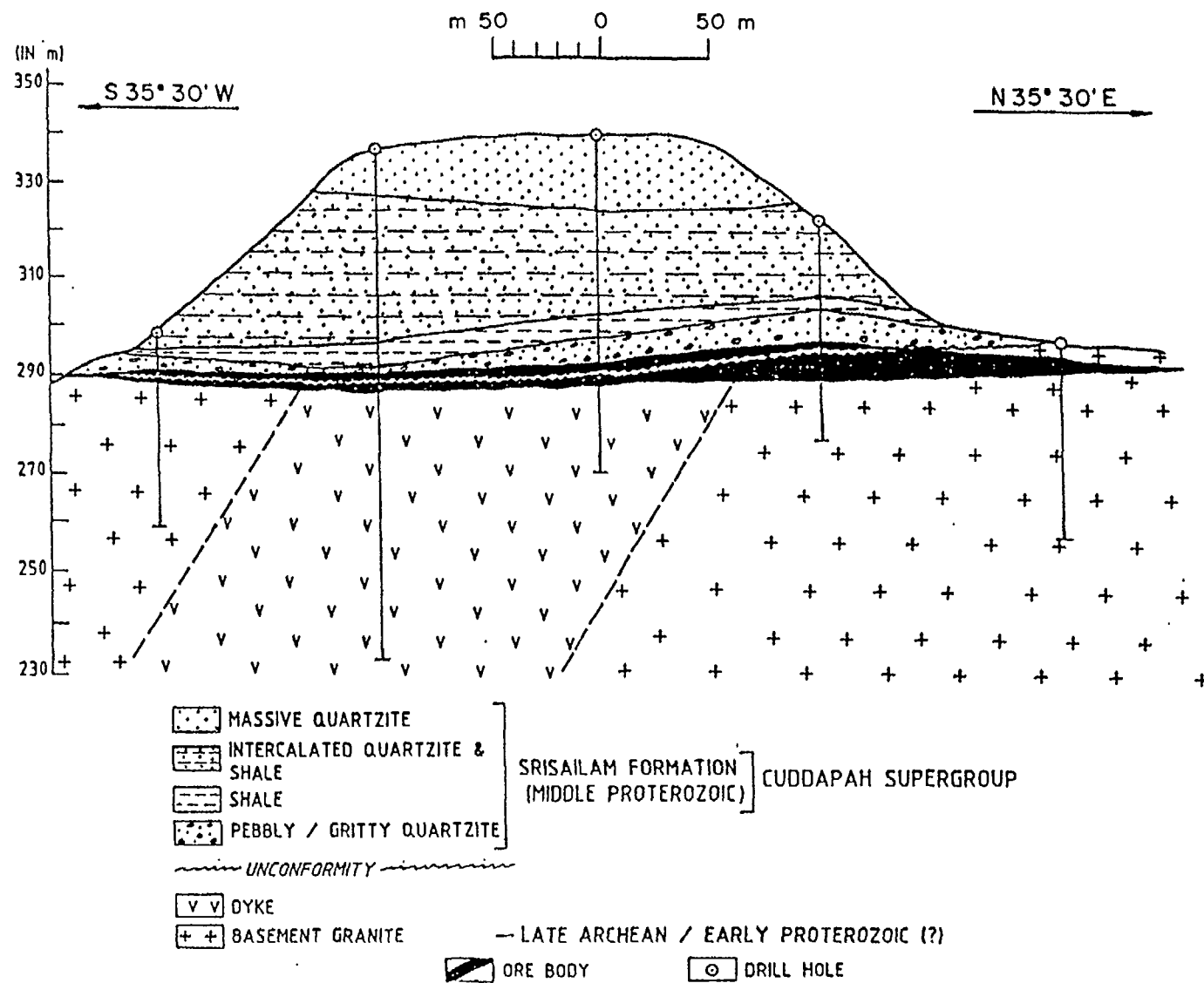


FIG. 7. Geological section across Lambapur outlier.

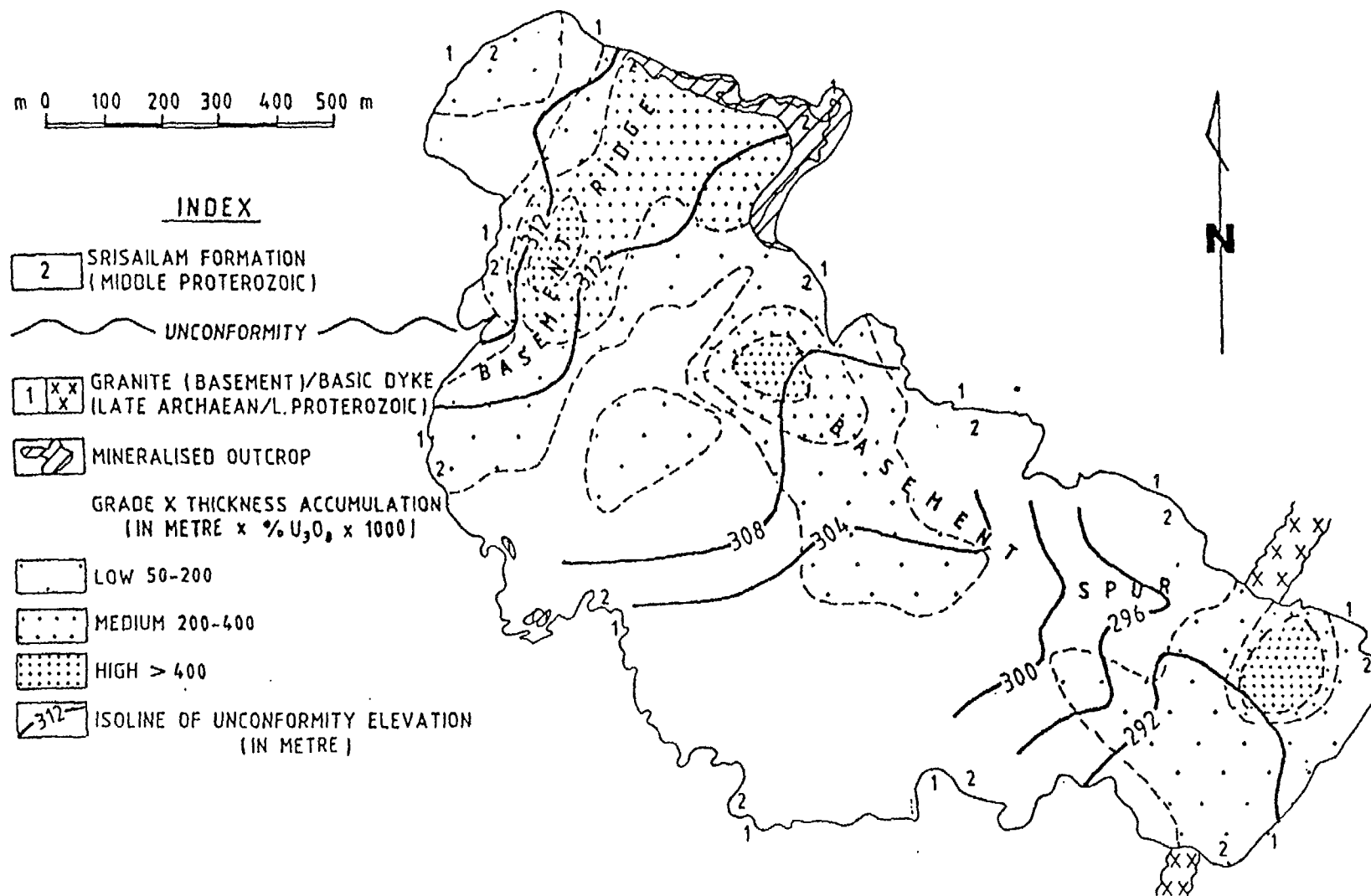


FIG. 8. Isoline map of U accumulations combined with isolines of unconformity elevation, Lambapur area, Andhra Pradesh.

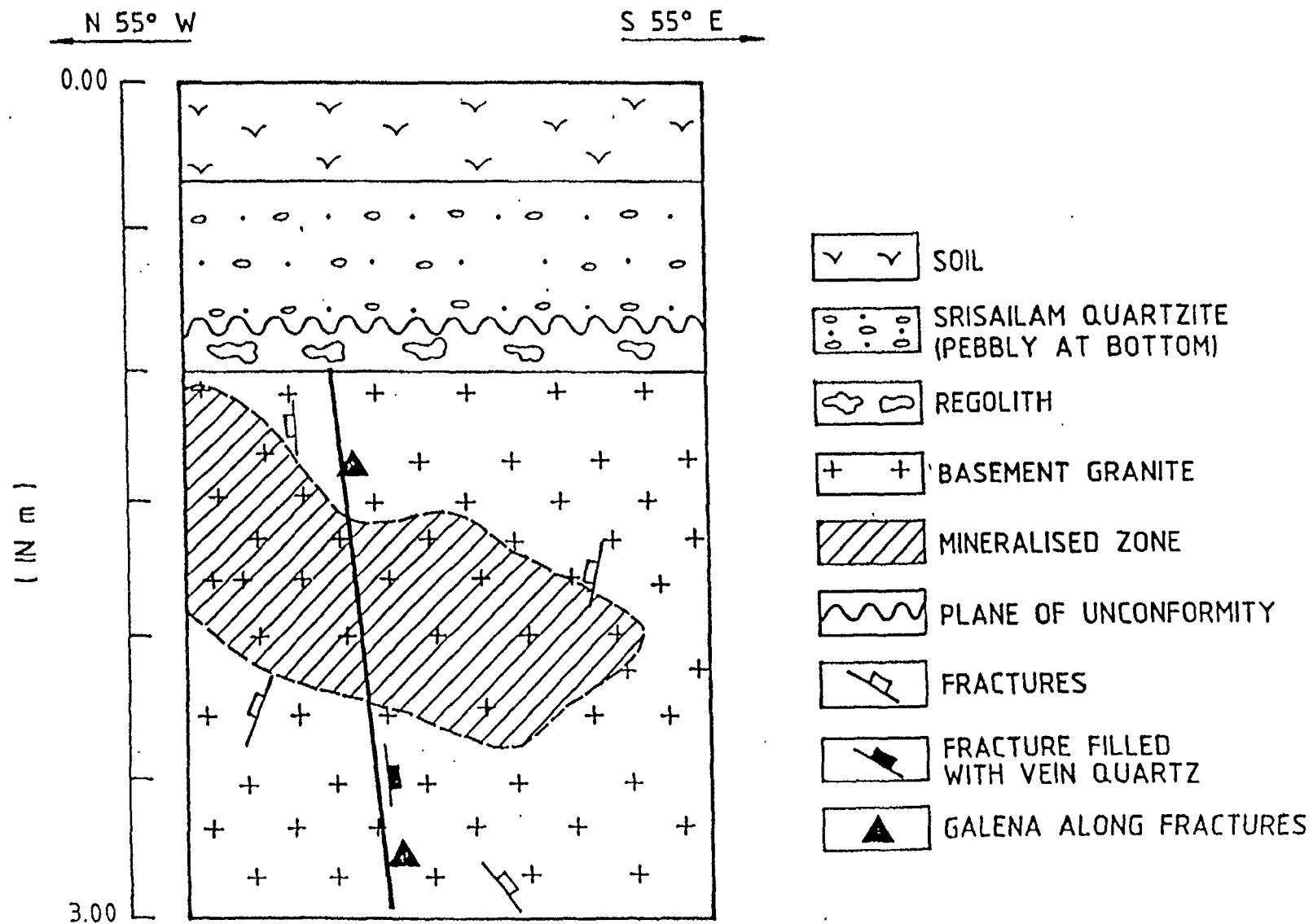


FIG. 9. Section of pit exposing unconformity and mineralization, lambapur area.

The nature and controls of uranium mineralization, established by drilling data, can be summarized as:

- (a) major part (more than 85%) of the ore body is confined to the basement granite below the unconformity;
- (b) the mineralization has shallow depth persistence (up to a maximum of 5 m) below the unconformity;
- (c) it is not in the form of a blanket along the unconformity, but occurs as linear pods with very sharp borders of rich mineralization, trending NNE-SSW and NW-SE;
- (d) the NNE-trending linear pod of rich mineralization in the northwestern part of the outlier coincides with a basement ridge.

These observations have also been corroborated in a shallow pit which exposes the unconformity, regolith and uranium mineralization in the granite (Fig. 9). Mineralization appears to be controlled by NNE-trending vertical fracture in the basement granite, filled with druzy quartz associated with kasolite ($\text{Pb}_2(\text{UO}_2)_2(\text{SiO}_4)_2 \cdot 2\text{H}_2\text{O}$) and galena. Such an association clearly points to open-space filling and hydrothermal nature of mineralization at Lambapur.

Thus, it is evident from the drilling data that two prominent sets of fractures/faults in the basement granite, trending NNE-SSW and NW-SE, have played a major role in concentrating uranium at Lambapur. The intensity of fractures within the granites and their intersections with the unconformity, thus apparently control the loci, shape and grade of mineralization. The role of basic dyke in generating necessary geotherms, as established in the Athabasca Basin, Canada[3], is yet to be properly understood.

3.4. Petrography of the host rocks and mineralogy

The host rock for uranium mineralization at Lambapur is mainly granite, which is medium- to coarse-grained and composed of phenocrysts of microcline-microperthite, orthoclase-microperthite, oligoclase and quartz. Microcline-microperthite shows post-crystalline deformation with fractures hosting clusters of chlorite, sphene, epidote etc. and uraninite with secondary development of uranophane [14].

The host Srisailem arenite varies from feldspathic sandstone to arkose. The alkali feldspars are as high as 40% in some samples. Uraninite and secondary uranophane are hosted by fractures within these arenites.

The host dolerite dykes are highly altered and consist of plagioclase and chlorite (altered from pyroxene/amphiboles) with minor calcite, epidote, sphene, apatite, biotite and some opaque minerals. Discrete pitchblende, with typical colloform texture, was first identified from the dolerite dyke.

XRD studies on sludge cuttings from borehole and on the ore obtained from a shallow pit have confirmed the presence of uraninite in association with galena and secondary uranium minerals uranophane and kasolite.

4. ORE RESERVES

At Lambapur 76 percussion boreholes were drilled to explore the entire outlier. Out of these, 44 boreholes (56%) encountered positive intercepts with grades ranging from 0.060 to 0.300% U_3O_8 .

The ore-reserves estimates have been attempted both by variable area of influence (plan) and sectional methods, and based on the results obtained from 44 percussion boreholes, it is tentatively of the order of 1600 tonnes U_3O_8 of 0.10% U_3O_8 grade. The reserves are provisional because the data from coring boreholes is only partially available. It is significant that the linear trends of mineralization established by percussion drilling have been confirmed by infilling core drilling and grades encountered are mostly higher than the non-coring boreholes.

5. COMPARISON WITH CLASSICAL UNCONFORMITY-TYPE DEPOSITS OF CANADA AND AUSTRALIA

The Proterozoic unconformity-type deposits, identified in Canada and Australia, are low-cost, high-grade and have large resources. These deposits are of two classes: (a) fracture bound, and (b) clay bound, and are defined by their setting with respect to the unconformity, host rock and ore-grades [2]. The principal recognition criteria for this class of deposits are defined on the basis of their salient geological features and are based on parameters such as host environment, wall-rock alteration, mineralization and age constraints.

The Lambapur uranium occurrence is in the early exploration stage and many of the geological characteristics are yet to be established. Two most fundamental criteria namely (i) the presence of Lower Proterozoic pelitic carbonaceous sediments, and (ii) the age of the unconformity, are not yet established. The major similarities and dissimilarities recognized are discussed briefly.

5.1. Similarities

The major similarities recognized include :

- (a) the age of the basement granite, established at 2268 ± 32 Ma to 2482 ± 70 Ma [Pandey et al., 1988]. The granites around Lambapur are anomalously rich in uranium (mean 27, range 9 to 53 ppm; $n = 170$ and thorium (mean 35, range 11 to 61 ppm; $n = 17$) with U/Th ratio of 0.77.
- (b) The Srisaillam Formation, unconformably overlying the basement granite, and occurring as cover rocks for the Lambapur uranium occurrence, is of Middle Proterozoic age.
- (c) Sharp linear trends of uranium mineralization are probably controlled by basement fractures.
- (d) Association of pitchblende/uraninite with galena, chalcopyrite and pyrite.

5.2. Dissimilarities

The most striking dissimilarity is the host rock for uranium mineralization, which is a sheared/fractured granite at Lambapur, in contrast to the pelitic-carbonaceous schists and gneisses in both the Athabasca Basin, Canada and the Pine Creek Geosyncline, Australia. This aspect probably accounts for much lower uranium grades encountered at Lambapur.

6. POTENTIAL AREAS IN THE NORTHERN CUDDAPAH BASIN

6.1. Srisaillam sub-basin

The geological set-up at Lambapur represents the geological set-up of entire Srisaillam sub-basin which has an areal extent of more than 3 000 km². Therefore, the southward continuation of

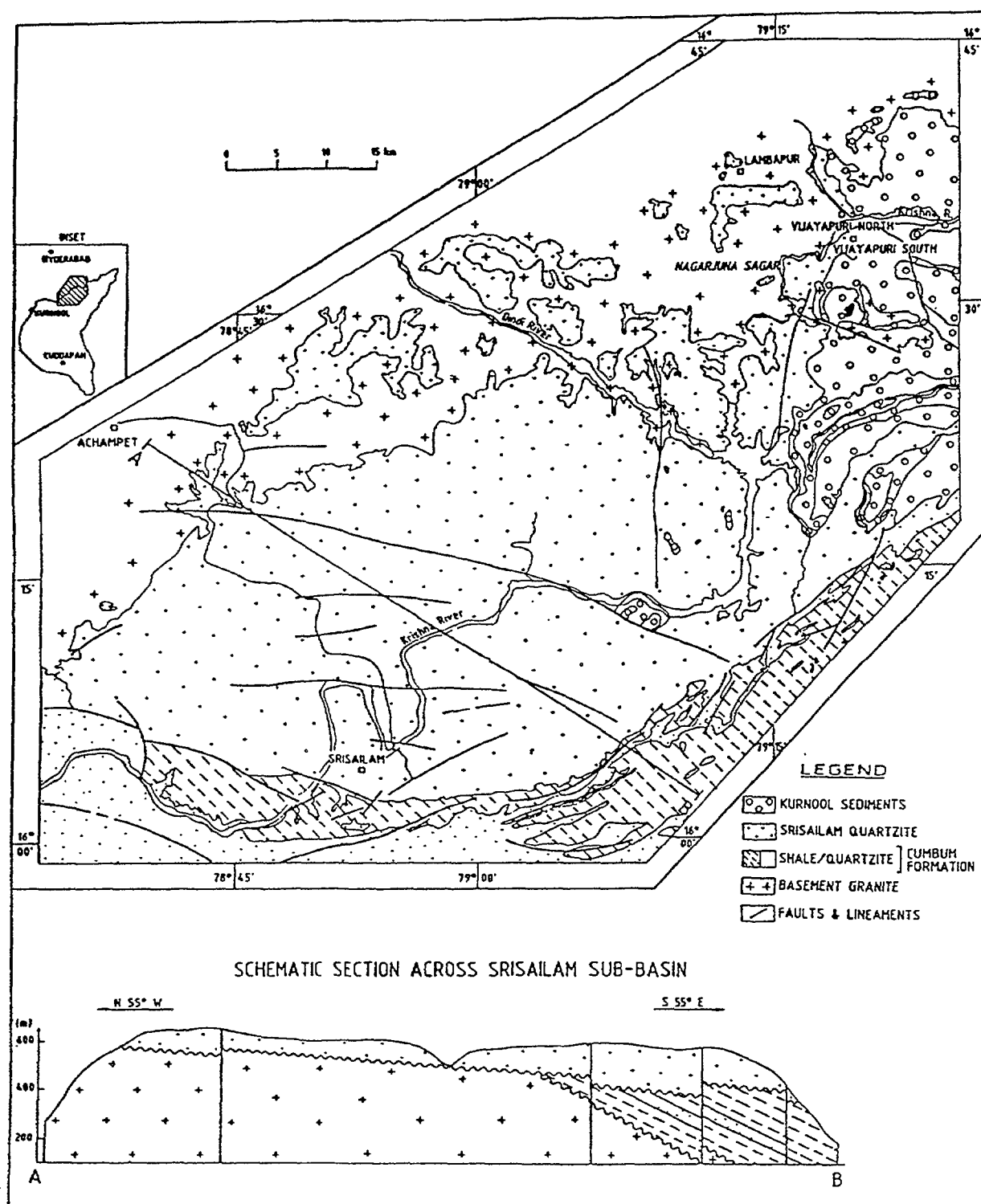


FIG. 10. Geological map of Srisailam sub-basin.

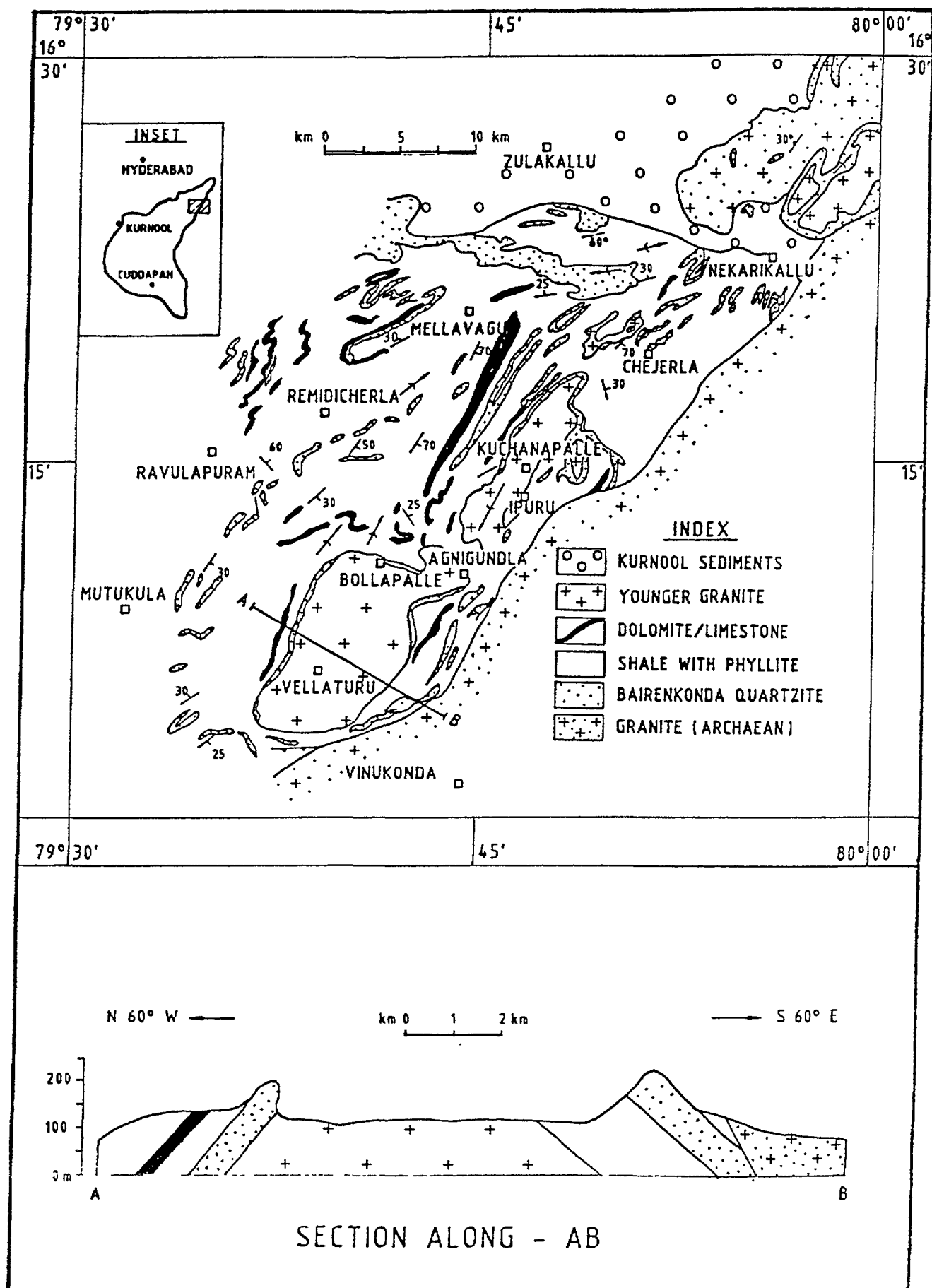


FIG. 11. Geological map of part of eastern margin of Cuddapah Basin.

the lithostructural set-up seen at Lambapur, into the main Srisailam sub-basin and outliers on its fringes, may host many such medium-grade occurrences/deposits hosted in fractured granite below the cover rocks.

In this light, numerous uraniferous anomalies, located since 1990 in different Srisailam outliers, therefore, gain considerable significance. The two most prominent outliers of Srisailam Formation at Yellapur and Chitrial are located to the south and south-west of Lambapur (Fig. 10). Although the outcrop uranium anomalies are of comparatively lower order as compared to Lambapur, these two outliers form promising exploration targets owing to their dimension. The Yellapur outlier is presently being explored by large interval non-coring boreholes and a number of significant results have already been obtained, thereby bringing exploration of other outliers and the main Srisailam sub-basin on much stronger grounds. Apart from the Srisailam sub-basin, other areas which could be potential for such mineralization is, probably, the Cumbum Formation of the Nallamalai Group in central and north-eastern parts of the Cuddapah-Basin.

6.2. Areas of Nallamalai fold belt

The Cumbum Formation of the Nallamalai Group are dominantly argillaceous and have good development of carbonaceous shales and phyllites. Being part of the Nallamalai fold belt these are intensely folded with major NNE axial trends. In the northeastern margins of the Nallamalai fold belt, large exposures of granite in three domal structures, namely, the Nekarikallu, Ipuru and Vellaturu domes, within the Cumbum Formation, are considered either as intrusive or as reactivated basement which have dommed-up the sedimentary pile (Fig. 11). The Agnigundala leadcopper belt, hosted by the dolomites and dolomitic limestones interbedded with the chloritic and carbonaceous phyllites of the Cumbum Formation, occur along the closure of the Vellaturu dome [19]. Such an association is well established in the Rum jungle and South Alligator Rivers Uranium Fields, Australia which have overlapping Ag-Pb-Zn-Cu and uranium zones [5].

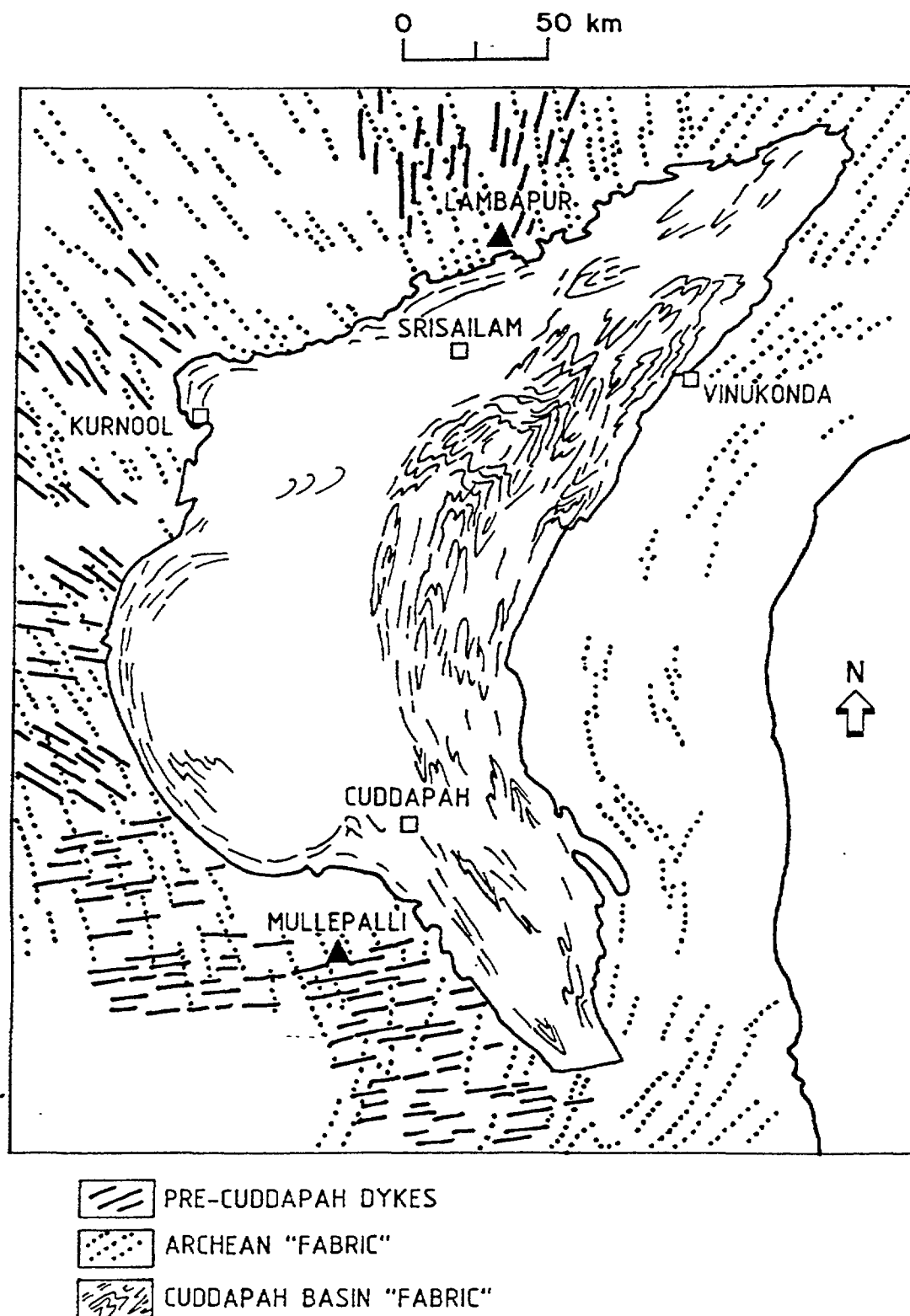
The continuation of such a set-up under the Srisailam Formation or younger cover rocks, presents an ideal geological set-up for hosting the classical unconformity-type deposits in the northern parts of the Cuddapah basin. This needs detailed scrutiny and exploration as part of the overall strategy in Cuddapah basin.

7. DISCUSSION

Renewed surveys taken up in the northern parts of the Cuddapah basin, to locate mineralization/deposits similar to the classical unconformity-type at the unconformity between the basement and the Middle-Proterozoic metasedimentary rocks have revealed the presence of Lambapur uranium occurrence.

Discovery of Lambapur occurrence, and many such occurrences, associated with granites, dykes and Srisailam Formations along the margins of the numerous dissected outliers of Srisailam Formations, have opened-up new exploration scenario in the northern parts of the Middle Proterozoic Cuddapah Basin. The influence of the unconformity and the fractures in the basement granite on controlling the uranium mineralization is now well recognized. It makes the Lambapur occurrence a unique deposit, akin to the classical unconformity-type deposits of Canada and Australia, except for its medium grades. Efforts in the identical geological set-up of the Srisailam Sub-basin would result in establishing many such deposits of similar grades.

The Lambapur occurrence, although of medium grade and smaller size, could prove to be economically viable low-cost uranium deposit owing to its nearness to surface, favourable topography and gentle dip making feasible low-cost open-cast mining, ideal host rock allowing for good amenability to leaching and highly favorable infrastructure in the form of good approach and proximity to one of the major cities of India, i.e., Hyderabad.



MODIFIED AFTER VENKATKRISHNAN AND DOTIWALLA (1987)

FIG. 12. Tectonic fabric of Cuddapah Basin and its basement.

Numerous anomalies associated with the fractures in the basement granite, outside the margins of the Cuddapah basin, such as at Penevella, Mullepalli etc, could probably represent remnants of Lambapur-type mineralization, now exposed owing to the removal of the cover rocks (Fig. 12). The mineralized dykes at Lambapur and many other localities constitute part of the pre-Cuddapah dyke swarms which occur in three main groups: with NNE-SSW, NW-SE and ENE-WSW trends. The large swarm of N10°–15° E trending lineaments north of Srisailam (around Lambapur) are probably related to the NNE-SSW trending dyke swarms [18]. The reactivation of these lineaments during the Eastern Ghat orogeny probably led to the shearing of these dyke swarms and resultant uranium mineralization within them.

The lithology, probable age and metallic associations of the Cumbum Formation present an ideal geological set-up under the cover of the Middle Proterozoic (?) Srisailam Formation. The search for locating the classical unconformity-type deposit has, therefore, to be directed towards mapping those areas in the northern parts of the Cuddapah Basin which have all the favourable features for such a geological set-up to exist.

Recent surveys in areas of the Palnadu Sub-basin, exposing younger Kurnool Group sedimentary rocks, located to the east of Lambapur, has also indicated uraniferous anomalies with values upto 0.084% U_3O_8 . The anomalies are within the Banganapalle quartzite, the basal member of Kurnool Group, which directly overlies the basement granite as at Lambapur. Thus, the younger Palnadu Sub-basin has also become a promising target area of exploration.

8. CONCLUSION

In the course of our efforts to locate unconformity-type uranium deposit in this part of the Cuddapah Basin, the Lambapur-type of uranium mineralization has been identified in a similar geological setting showing features such as mineralization controlled by fractured basement, close to the unconformity and a thick sequence of cover rocks. Data presented in this paper also show that the Lambapur occurrence, though has a potential to develop into a significant medium-grade deposit, yet differs in some selected features from the classical well known unconformity-type. It is therefore felt that Lambapur-type unconformity related occurrence/deposit could be considered as a type area candidate for a sub-class under the classical unconformity-type uranium deposit. This could perhaps enable exploration activity to diversify into such Proterozoic basins, hitherto not taken-up for want of the features associated with the classical type.

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THE SYSTEM OF GEOLOGICAL EXPLORATION FOR POSSIBLE ISL INFLUENCE EVALUATION IN THE SOUTHWESTERN FOREFIELD OF THE STRÁŽ URANIUM DEPOSIT IN THE NORTH BOHEMIAN CRETACEOUS, CZECH REPUBLIC

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Abstract

The production of uranium ores in the area of North Bohemian Cretaceous significantly influenced the regime of groundwater and its quality. Since 1990, environmental impact assessments of uranium production has rapidly changed. From analysis of this impact, it can be stated that the large amount of contaminated water from ISL is the greatest potential risk. During 1992–1994, a lot of geological exploration was done in the area south-west of the ISL plant (the direction of natural groundwater flow). These works should answer, if the geological situation in this area has some risk elements, which could transfer the contamination risks from the Cenomanian aquifer to the surface. A complex of exploration methods, between synoptic and detailed scales, was set for qualification and quantification of potential risks. This complex should answer the given questions in a very short period. These methods have a logical progression. In the first phase, synoptic methods were used (satellite photos, airborne geophysics). At the same time, the re-interpretation of older geological and geophysical data was performed. Also the works for the regional hydraulic mathematical model of groundwater flow began in the whole area of North Bohemian Cretaceous. Based on the results of this work, surface geophysical and geological exploration started. It was first applied to the areas presenting the maximum of risk (faults with significant vertical movement along them). Results of this exploration were gradually put into the regional hydraulic mathematical model. This model was also targeted at the highest risk areas. So far, the first phase of qualification and quantification of risks resulting from the geological structure has been completed. The results from 1992–1994 signal the necessity of stabilizing contaminated groundwater in the production area. This fact determines the next remediation concepts e.g. the stabilization of groundwater in the production area and followed by gradual steps (e.g. minimal recovery of solutions for the depression cone development in the contaminated area, time-spatial course of contaminated water recovery, preliminary setting of limits for contaminants content etc.).

1. INTRODUCTION

Uranium production using ISL in the area of North Bohemian Cretaceous has been the subject of increased professional public and state interest since 1989. At the beginning of 1992, the Czech government published decision No. 366192, which concerns only ISL on the Stráž deposit. The following topics are included:

- the principles of the ISL special regime on the Stráž deposit during 1992–1994 (restrictive production regime with running control),
- a programme of technical measures to begin removing of ISL's negative environmental influences,
- a programme of research and verifying work to determine the optimal course for remediation of the environment, which has been very strongly influenced by the ISL method on the Stráž deposit.

Geological exploration in the south-western forefield of the Stráž deposit has been performed within the framework of research and verifying work. This area lies in the direction of natural

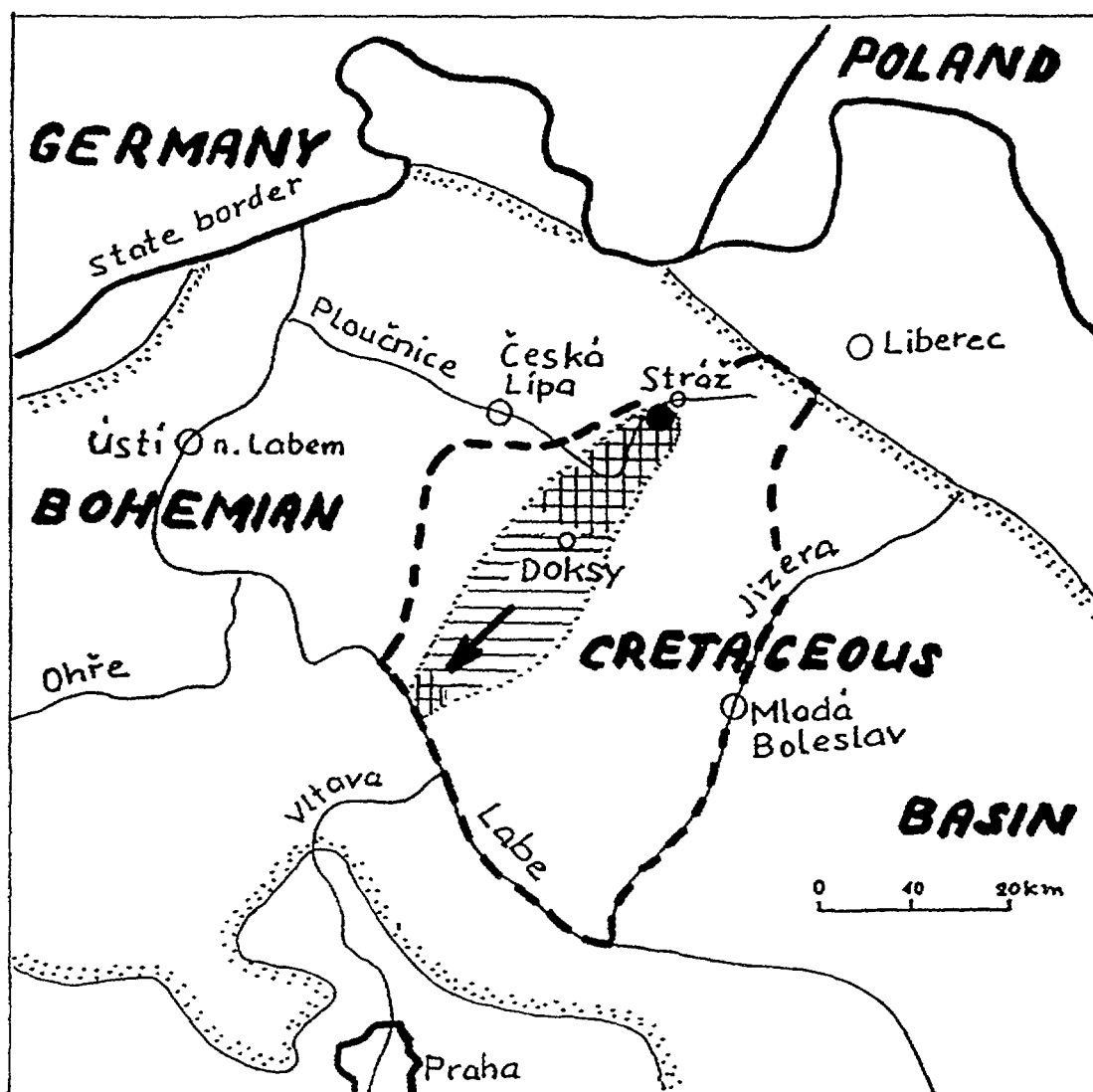


FIG. 1. Situation scheme of groundwater potential contamination risks in Bohemian Cretaceous Basins from ISL on the Stráž deposit.

groundwater flow from the ISL area. This work should answer, whether some geological elements can be considered as risks in connection with the existence of a large amount of contaminated groundwater on the Stráž deposit.

2. BASIC GEOLOGICAL AND HYDROGEOLOGICAL CHARACTERISTICS OF THE AREA

The area of ISL on the Stráž deposit is situated on the northern margin of the Bohemian Cretaceous Basin in its partial structural-geological unit called the Stráž block. (Fig. 1). The complex of Upper Cretaceous sediments, which is defined by the boundary of the regional hydraulic model (see Fig. 1), can be simply characterized as the part of the geological structure with two main aquifers and one aquiclude between them.

The lower aquifer is associated with Cenomanian sandstones, which are underlain by non-permeable paleozoic and precambrian rocks. The Cenomanian aquifer is confined. In the area of the Stráž block, many of the uranium sandstone-hosted deposits occur. Almost 200 million m³ of contaminated water from ISL uranium production are contained in this aquifer. The northern margin of the Cretaceous Basin is an infiltration area and the line of the Labe River is considered as the final discharge area. The general slope of the groundwater level in the cenomanian aquifer is from NE to SW.

The upper aquifer is situated in Middle Turonian sandstones and their thickness depends on the surface morphology. This aquifer has a free surface and is soaked by the infiltration of precipitation in its whole area. The discharge of this aquifer is locally directed to water streams, generally from NE to SW.

The aquifers are separated by lower Turonian layers with argillaceous development, which has the character of aquiclude. The thickness of the Upper Cretaceous sedimentary complex in the area of interest is between 150 and 350 m. This area of interest can be defined in the north-western part of the area of the regional mathematical model. (Fig. 1) We need to add that the sunken Tlustec block adjoins the north-western border of the regional hydraulic model. Cretaceous sediments in this block have a thickness of up to 700 m. The upper Turonian and Coniacian layers in the Tlustec block neighbour middle Turonian of the Stráž block and create the stratigraphically highest unit. This unit functions as an aquifuge in its lower part and a non-important aquifer in its upper part. The whole area was affected by Tertiary volcanism, whose intensity increases to the west.

3. PROBLEM SOLVING LIMITATIONS

As problem solving limitations, which influenced the systematic arrangement of exploration we can consider:

- (a) demands of the final exploration output
- (b) the deadline for the solutions
- (c) financial funds
- (d) the initial knowledge of the geological and hydrogeological conditions
- (e) availability of exploration methods

ad (a) Demands on the final exploration output are summarized in the following:

- acquiring more accurate structural-geological knowledge in the area of interest;
- determination of geological risk elements;
- acquiring more accurate knowledge of the hydrogeological conditions in the area of interest, based on additional geological information, verifying technical works (boreholes, hydrodynamic and trace tests, hydrological balances etc.);

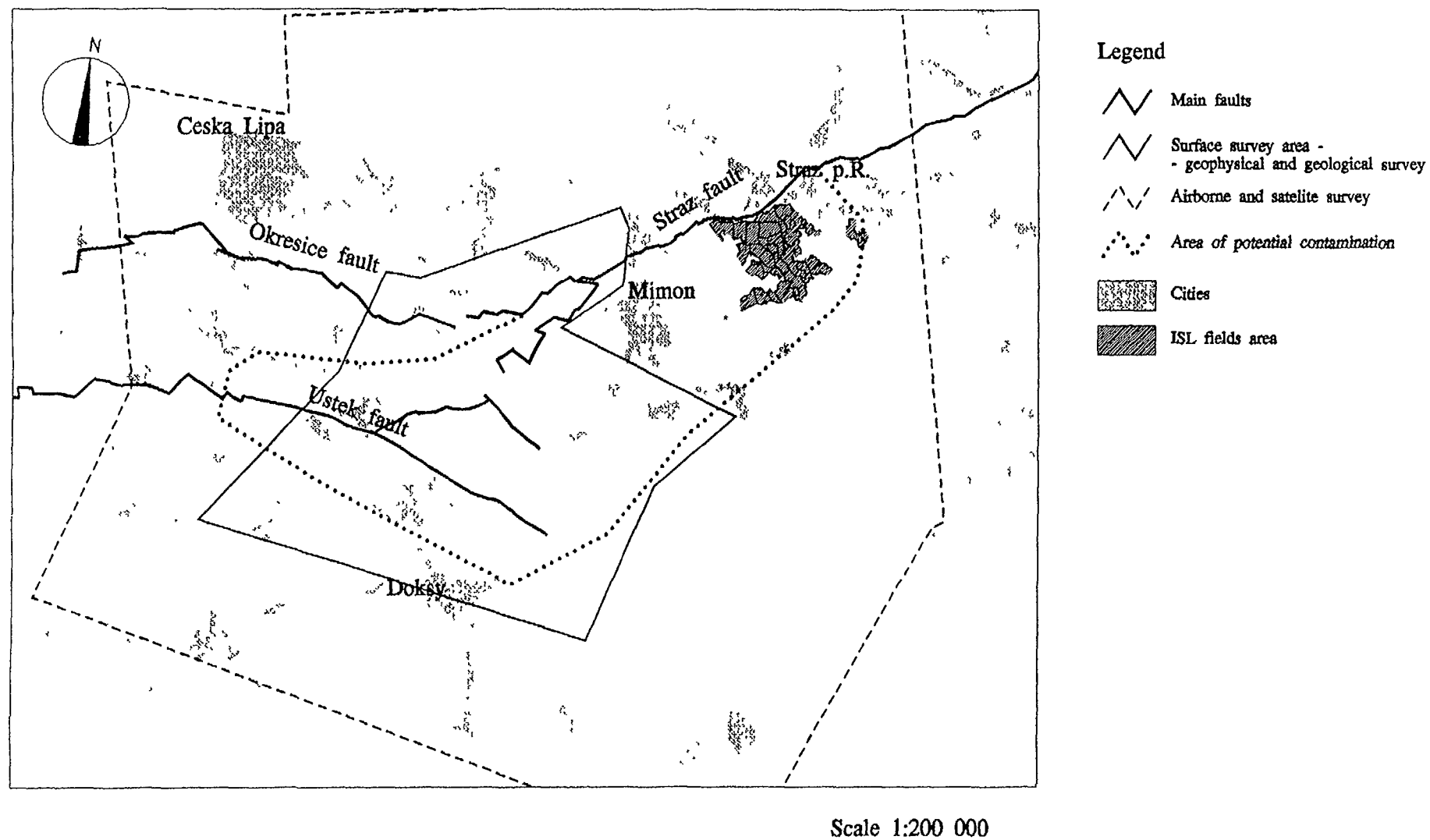


FIG. 2. Survey of the area of interest related to ISL mining — period 1992–1994.

- variant calculations of mathematical hydraulic and hydrochemical modelling (spreading of contamination from the ISL area, overflow of contamination to the upper aquifer etc.).
- ad (b) The deadline for problem solving was very strict, because the results of performed exploration had a direct connection with planning of the whole remediation concept of the environment after ISL mining in Northern Bohemia. Work started in mid-1992 and the main results should be known before the end of 1994 in order to use them for the above mentioned decision making strategy in the North Bohemian production area. In 1995, exploration will be aimed at more accurate and detailed work and at the complex processing of its results.
- ad (c) The financial funds for exploration represented approximately US\$ 900 000 for the period 1992–1995. More than 90% is covered by the Ministry of Economics the remainder by DIAMO. By the end of 1994, about 70% will have been spent.
- ad (d) The initial knowledge of geological and hydrogeological conditions (Fig. 2) were different in individual parts of the area of interest. The highest level of knowledge is in the area of ore deposits, the lowest level is in the former military area situated SE of Mimon.
- ad (e) A sufficient number of domestic companies existed and provided a good range of necessary methods for problem solving.

4. ORGANIZATIONAL ARRANGEMENT AND CO-ORDINATION OF EXPLORATION

DIAMO was the main organization for solving exploration problems. DIAMO has a lot of experience in geological exploration within the boundaries of the Bohemian Cretaceous Basin. It also has extensive databases of geological information, verified mathematical models of the groundwater flow from the production area, verified hydrochemical models from ISL etc. On the other hand the uranium production contraction programme also reduced DIAMO's geological activities (e.g. abolishing its exploration company, etc.).

The proper organizational arrangement is as follows:

DIAMO Stráž Pod Ralskem:

Management of work, its co-ordination, hydrogeology of production areas, hydraulic and hydrochemical modelling in production areas, technical works (boreholes etc.), laboratory works, the complex processing of results

AQUATEST Praha:

Major supplier of regional hydrogeological works (hydrology, hydrogeology, hydraulic and hydrochemical modelling, pumping tests, trace tests, etc.)

RADIUM Liberec:

Major supplier of geological and geophysical surface survey, digitalization of map basis and results.

Other suppliers of partially specialized subdeliveries are:

Airborne survey — (*GMS Praha*)

Thermometric methods — (*BGC Praha*)

Analysis of satellite prints (photos) — (*VM Consult Praha*) and others.

The guarantee of professionalism and financial funds are ensured by the Czech Ministry of Economics and by the commission of specialists, appointed by the Czech Minister of Environment. They oversee fulfillment of all tasks stemming from the Government decision about ISL (No.366/92).

5. THE SYSTEM OF EXPLORATION SOLUTIONS DURING 1992–1994

Exploration of wider areas of interest around ISL represent atypical, purposely targeted work complexes characterized by:

- a wide spectrum of exploration methods, which also serve for obtaining partial solutions towards eliminating problems;
- partial permeation of exploration stages (in the sense of classical geological exploration) ~ which is called for by the high dynamism of problem solving over a large area (more than 500 km²);
- a combination of synoptic exploration stages, detailed exploration methods, special exploration methods, re-interpretation of older geological exploration results and mathematical modelling;
- running interaction among all the performed exploration methods;
- high flexibility of problem solving;
- ongoing use of partial results for precisising whole remediation concepts within DIAMO's production area;
- a high use of obtained results for other activities, which are not directly connected with the solved problems.

5.1. Exploration in 1992

Three main priorities were followed during the exploration management.

- creating a mathematical model of groundwater flow in the whole region;
- rapid accumulation of synoptic information about the structural-tectonic conditions in the surroundings of ISL using "remote sensing methods";
- the beginning of surface exploration of the potentially endangered area.

The area lying above the stream of potential spreading of contamination from ISL with the groundwater level of Cenomanian aquifer higher than the groundwater level of Turonian aquifer can be considered as potentially endangered. Two potentially endangered areas were determined by mathematical modelling of the groundwater flow in the wider area. The first one is very distant, small and unimportant area near the Labe river. The second one is the main potentially endangered area of the Turonian aquifer between Stráž pod Ralskem and Doksy (see Fig. 1).

Setting of remote sensing exploration methods in the wider ISL area of interest (see Fig. 2) compensated the area exploration imbalance and speedily offered synoptic results. The main methods were structural-tectonic and exodynamic analysis of satellite and airborne stereoscopic pictures, airborne gamma spectrometry, magnetometry and airborne infrared sensing.

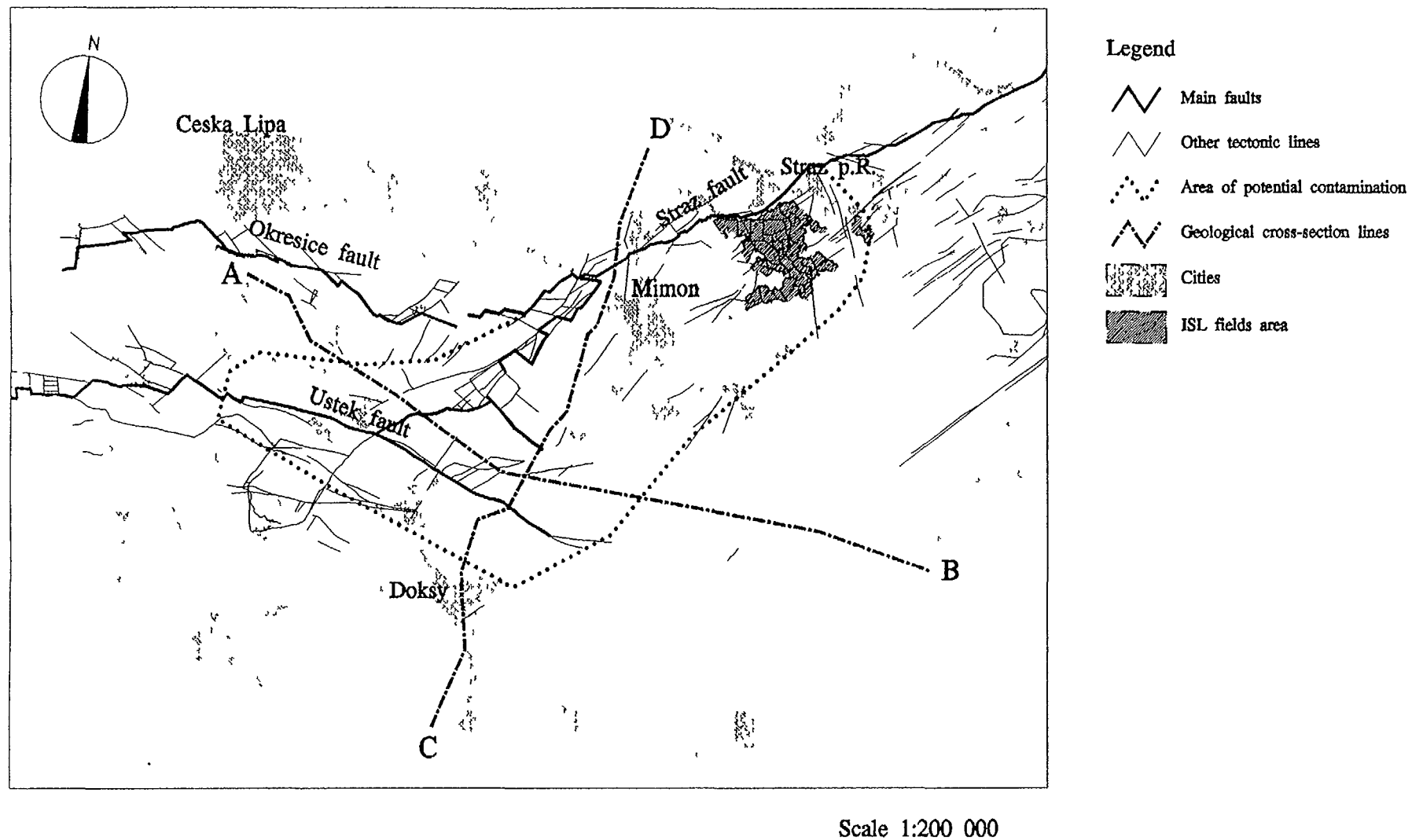


FIG. 3. Simplified structural-tectonic scheme of the area of interest related to ISL mining.

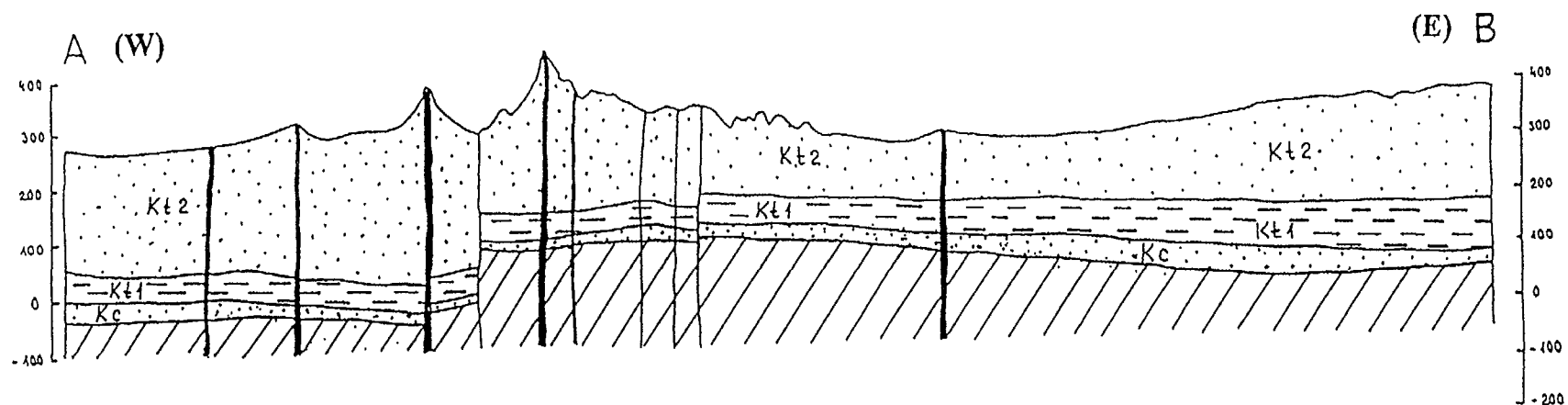


FIG. 4. Schematic cross-section.

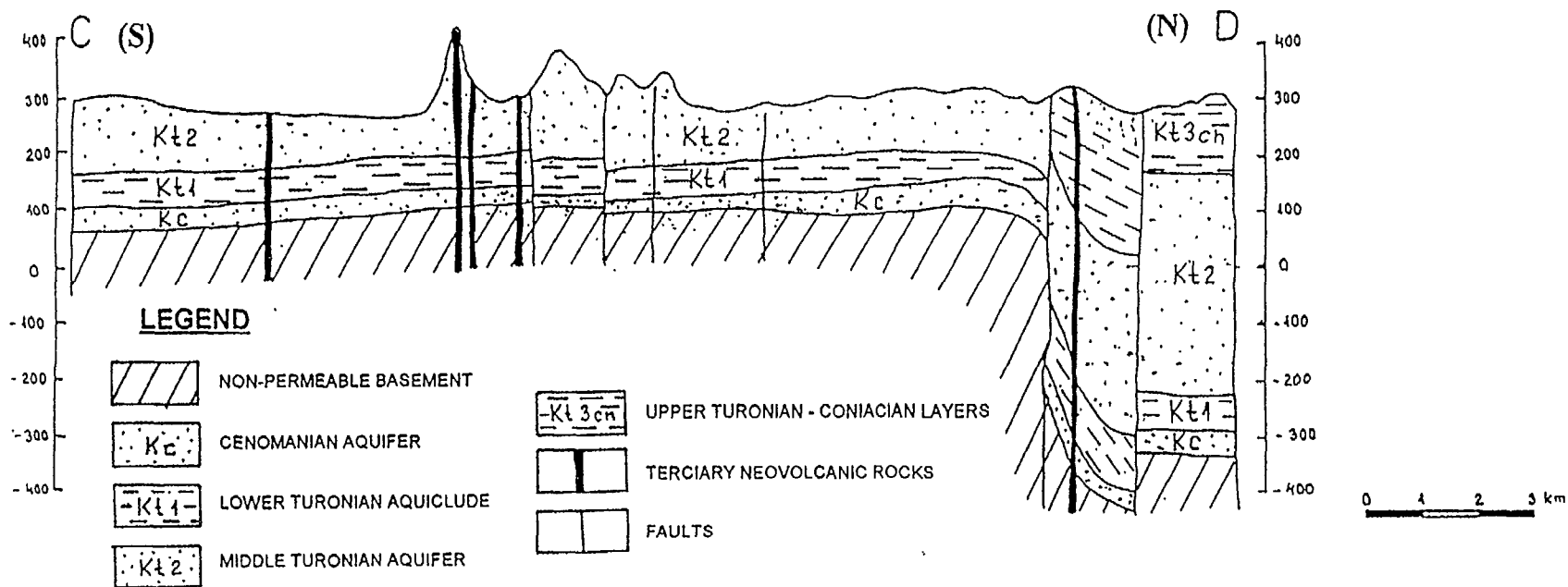


FIG. 5. Schematic cross-section.

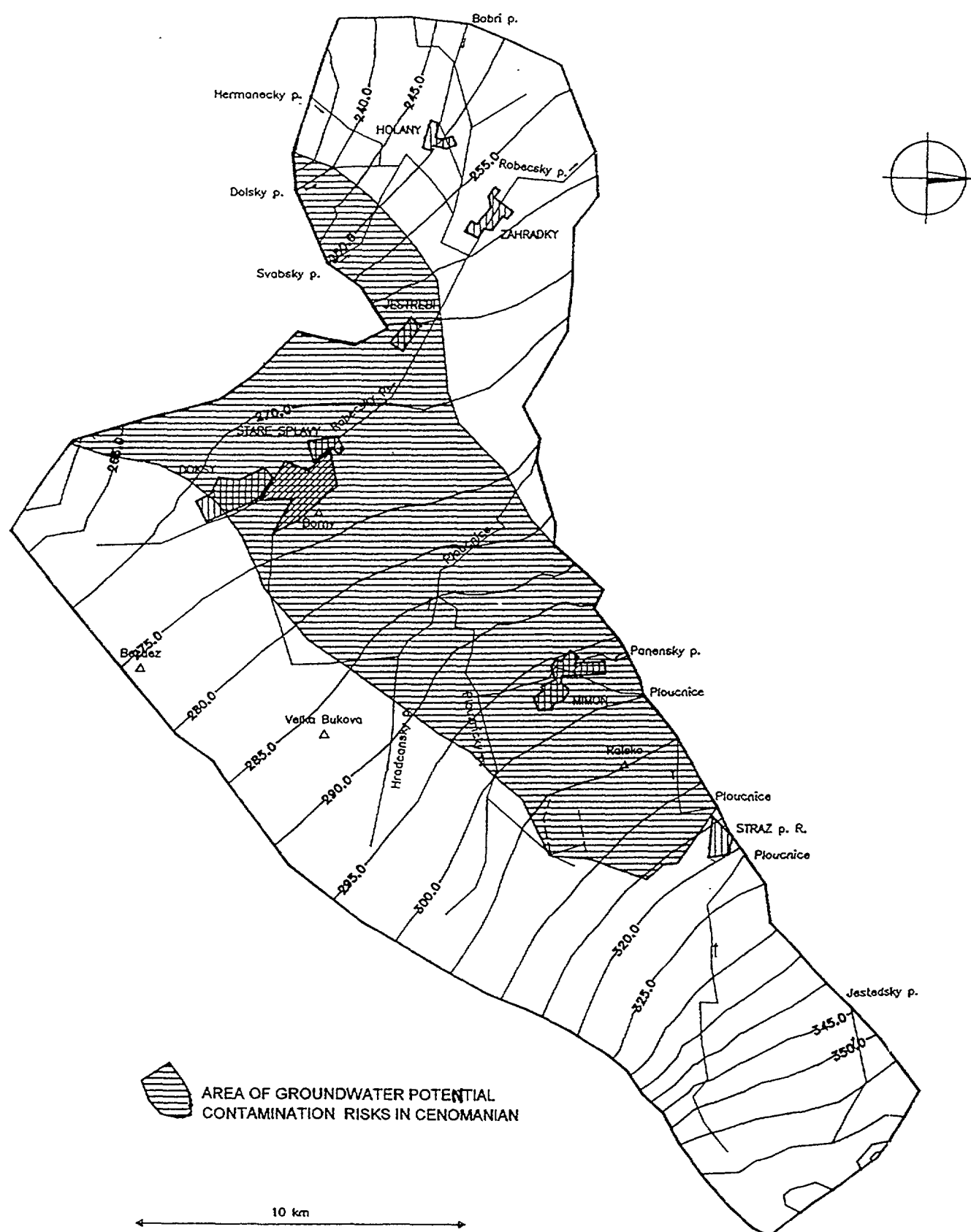


FIG. 6. Demarcation of groundwater potential contamination risks in Cenomanian aquifer.

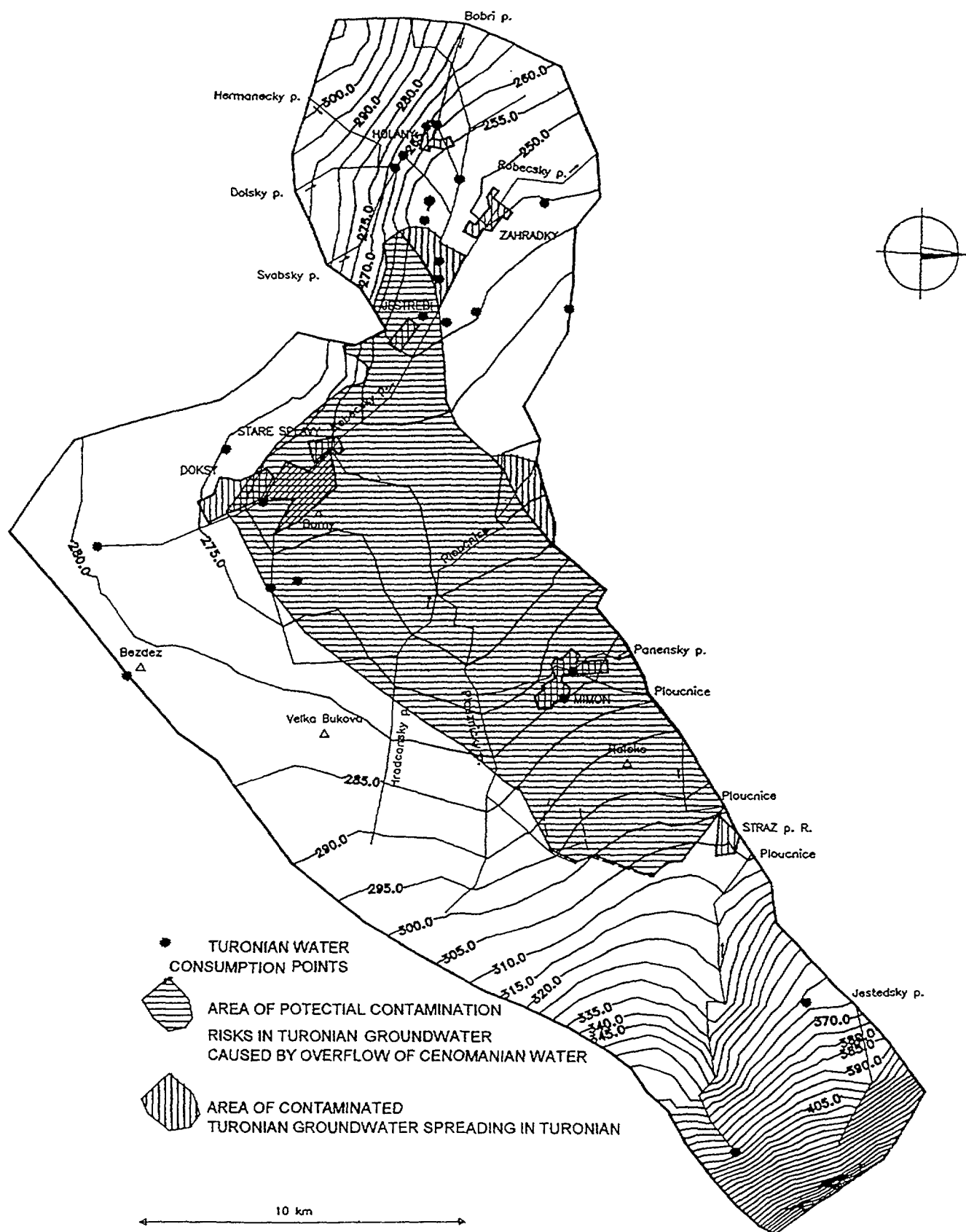


FIG. 7. Demarcation of groundwater potential contamination risks in Turonian aquifer.

Surface exploration started south-west of the production area, which was explored relatively well. In 1992, only the northern third of this surface exploration area was explored (see Fig. 2). Geoelectric and geomagnetic methods were used (line measurements).

5.2. Exploration in 1993

The main volume of work was targeted at surface exploration and hydrogeological work. Technical verifying works were also started (digging trenches, hydrogeological wells, etc.).

Surface exploration was completely aimed at the area of potential danger for the groundwater of the Turonian aquifer. This area was obtained from a mathematical model. Besides geophysical works, the classical geological mapping started. It was aimed at following the tectonic elements. The surface exploration covered about two thirds of the area (southern part — see Fig. 2). The structural-tectonic scheme was compiled at the end of 1993 (see Figs 3, 4 and 5).

Hydrogeological works were aimed at the establishment of a detailed mathematical model of the groundwater flow (the 2nd Generation Model), which could make a more accurate balances of endangered areas and flows between aquifers. On these two examples (see Figs 6 and 7) you can see the groundwater level after the Stráž deposit remediation finished. These examples show the remaining contamination spreading from ISL in the Cenomanian aquifer and its overflow into the Turonian aquifer.

In 1993, hydrogeological wells south of Mimon were drilled and one pumping test was performed.

5.3. Exploration in 1994

Exploration in 1994 is aimed at two important directions — surface exploration and hydrogeological works. The special method of thermometric exploration of the main faults is being performed in the area of interest. Major attention and effort are given to the digitalization of exploration outputs.

Surface exploration, which includes surface geophysics and geological mapping, moved back to the south-western part of the Stráž fault. It seems to be the riskiest area. Splitting of the Stráž fault in the area west of Mimon (see Fig. 3) creates a system of step normal faults, where direct contact between the Cenomanian and Turonian aquifers is present. Some of the riskiest areas were chosen for the application of specific exploration methods in order to further define the areas.

Hydrogeological works are aimed at the 3rd generation mathematical model of groundwater flow (the most detailed scale) and at the hydrochemical mathematical modelling of contamination overflow from the Cenomanian aquifer to the Turonian one. Conclusions for the contamination limits of groundwater after the completion of ISL on the Stráž deposit are drawn from these hydrochemical calculations. The stated calculations represent the basic input for the whole remediation concept in the production area, because they determine the time of necessary groundwater treatment on the Stráž deposit.

REMEDIATION PROGRAMME IN THE NORTH BOHEMIA REGION

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Abstract

The Bohemian Massif belongs to a very important uranium province where mining activities started in 1840. Since the end of World War II large mining operations have taken place resulting in a large accumulation of waste dumps and tailings. In 1969, the Stráž sandstone type uranium deposits were exploited by in situ leaching methods. 3.8 million tonnes of H_2SO_4 , 270 000 tonnes of HNO_3 and 103 000 tonnes of NH_4 were injected into the Cenomanian aquifer. It affects 188 million cubic metres of water and 28 km². The problem is compounded by the presence of an underground mine next to the Stráž deposit. The paper describes the environmental remediation programme established by the government. The present estimation of the time necessary for the decontamination of the Stráž deposit lies between 50 and 70 years.

1. HISTORICAL REVIEW

The Bohemian Massif belongs to a very important uranium bearing province. Uranium mineralization has been connected with Post-Variscan hydrothermal activity and emplacement of carbonatic dikes with uranium mineralization. Uranium ores have been mined in Jáchymov (Joachimsthal) since 1840 first for making paints and later when radium and polonium have been discovered by Mme Curie even for radium producing. During the years 1907–1939 a total of 2.5–5.5 g of radium per year have been produced there.

The years 1945–1960 started the period of exploitation of uranium ores for army purposes and the first Soviet atom bomb was manufactured from uranium ore of the Jáchymov district.

After World War II the uranium exploration grew rapidly as a large scale programme in support of Czechoslovakian uranium production industry. A systematic exploration programme including geological, geophysical and geochemical surveys and related researches, was carried out to assess the uranium potential of the entire country.

During the long period of underground mining especially since the end of World War II the devastation of landscape, by means of waste dumps accumulation, tailings and other workings, have been enormous. Subsequently all these activities have a negative impact on the environment including surface and groundwaters and soils.

In the mining districts of Jáchymov, Tachov, Horní, Slavkov, Příbram there are more than 38 large waste dumps and many small waste dumps. They originated from the extensive prospection carried out all over the Bohemian Massif.

Table I. shows main regions and volume and areas of land affected by waste dump deposition.

TABLE I. VOLUME AND AREAS OF LAND AFFECTED BY WASTE DUMP DEPOSITION

Region	Volume (m ³)	Area (m ²)
West Bohemia	10 662 000	641 000
North Bohemia	1 302 000	114 000
Příbram Region	28 511 000	1 299 000
Dolní Rožínka	2 623 000	406 000
Total	43 098 000	2 460 000

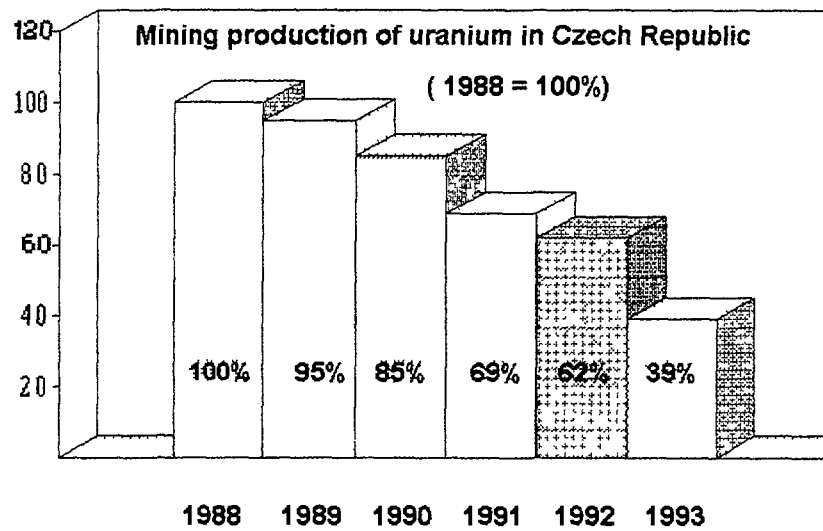


FIG. 1. The percentage of uranium mining production in Czech Republic.

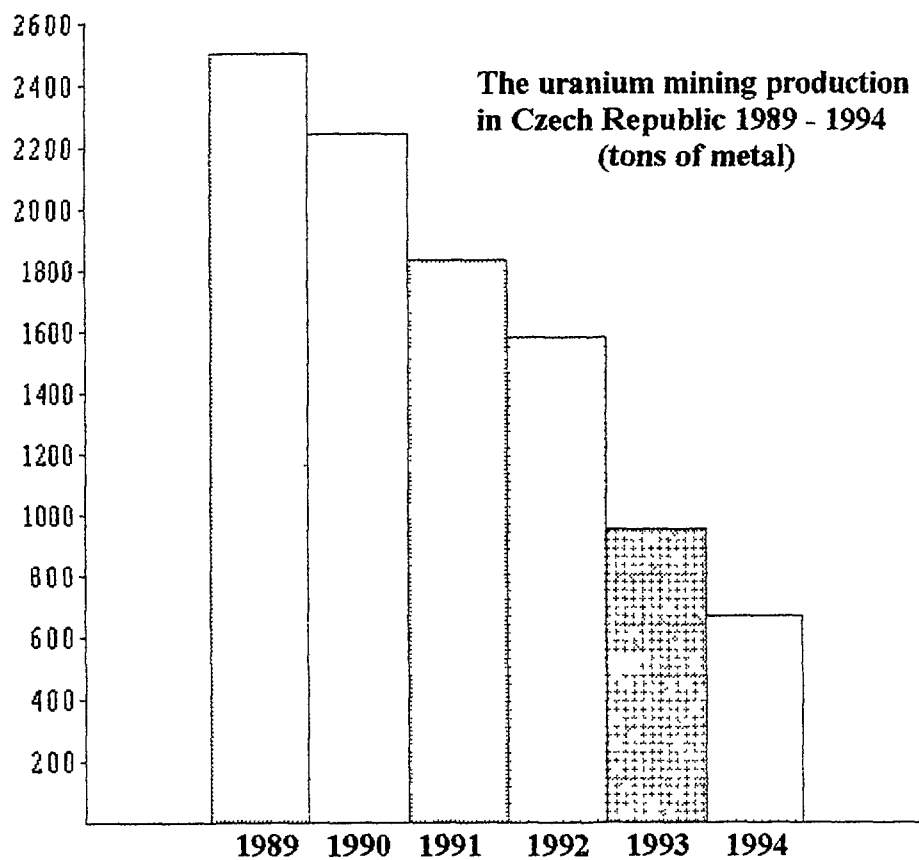


FIG. 2. The uranium mining production in Czech Republic 1989-1994.

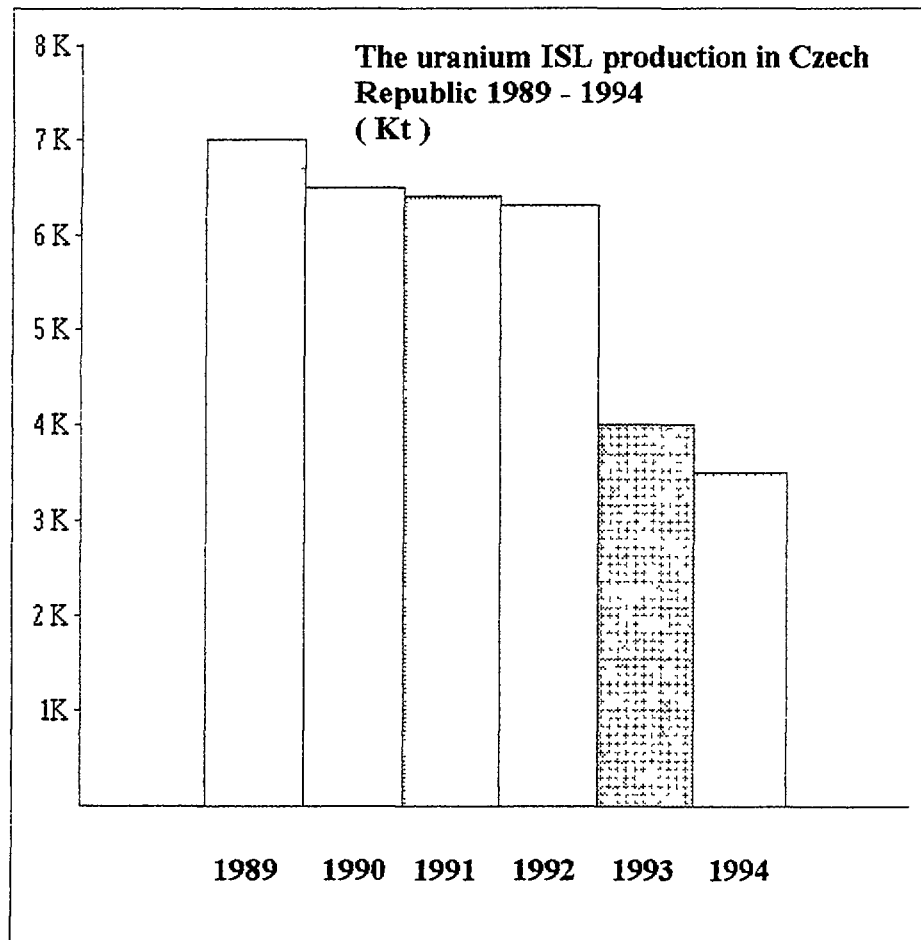


FIG. 3. The uranium ISL production in Czech Republic 1989–1994.

2. URANIUM MINES IN THE CZECH REPUBLIC

The majority of uranium deposits in the Bohemian Massif are of the vein type, however, this paper will discuss the stratiform, sandstone type which are situated in the North Bohemian Cretaceous basin. Until the year 1989 a total of 96 000 t of uranium has been produced in all mines in the Czech Republic (85 000 t from underground mines and 11 000 t from ISL fields).

The North Bohemian area with its stratiform sandstone type of uranium deposits is the newest ore producing district in Czech Republic and its exploration has started in 1969.

Because of the relatively large deposit (about 200 000 t U) the uranium production was meant to cover all the long term needs of nuclear power plant supplies, including export to former COMECON countries.

This situation has been changed dramatically after 1989 with the changes of the political and economical situation in Europe. The uranium spot prices in the world decreased rapidly and uranium mines were not able to compete on the world market at such a low price.

From the following Figs 1 to 3, describing the mining production in Czech Republic 1989–1994, the recession trend of mining production is clearly visible which originated after the collapse of the Soviet Union when a worldwide nuclear disarmament programme was started.

Government decisions have been taken to close down a majority of uranium mines and the subsequent assessment of negative impacts of uranium mining on the environment have shown the necessity to install the remediation programme for all areas where uranium mines had operated.

A complex evaluation of in situ leaching mining activity and its negative impact on the environment in North Bohemian cretaceous basin together with the development of environmental evaluations has been clearly stated in Government Decision No. 3661/1992 (ISL) and No. 429/1993 (underground mining) see below.

A special regime of mining was started for the period of 1992–1994. Its progress is evaluated and directed by Government Commission of Experts that designed a programme of scientific research and optimal ways of mining and restoration of Stráž pod Ralskem Mine and Hamr Mine. This Commission simultaneously controls the effectiveness of executed ecological programmes.

3. COEXISTENCE OF UNDERGROUND MINING AND IN SITU LEACHING OF URANIUM IN NORTH BOHEMIA

The main problem for uranium exploitation in this area is the coexistence of two large production complexes — classical deep mining on the Hamr deposit and ISL on the Stráž deposit. These deposits are situated near each other and they have negative influence on the geological environment of the whole area.

3.1. Geological setting

Uranium concentration in Northern Bohemia occurs within sediments of the Upper Cretaceous platform unit of the Bohemian Massif which are forming tectonic blocks. The most important concentration of uranium ores were found in s.c. Stráž block.

Cretaceous sediments have been deposited on metamorphosed basement consisting of low metamorphosed rocks and acid granitoids. Their sequences range from Cenomanian to Turonian. The Turonian sequence represents an important aquifer of drinking water.

The average thickness of the whole Cretaceous complex in the Stráž block is about 220 m. The basalt volcanics of tertiary age penetrate the Cretaceous beds.

The stratiform uranium mineralization is confined to the lowest part of Cenomanian beds (washout sediments) and has an unusual association of elements: U-Zr-P-Ti (uraninite UO_{2+x} , ningyoite, $(\text{CaU}(\text{PO}_4)_2 \cdot n\text{H}_2\text{O})$ and hydrozircon $(\text{Zr}(\text{Si}_{1-x}\text{O}_{4.4x}(\text{OH})_{4x}) \cdot n\text{H}_2\text{O})$

3.2. Methods of extraction of ore

Two methods of extraction have been applied within the Stráž block since the late 1960s:

- (a) Classical underground mining in Hamr mine and Břevniště. The room and pillar method is used in the Hamr mine, and a relatively stable depression in the Cenomanian aquifer has been formed during the long term drainage. The depression is kept up by pumping out the quantity of mine waters reaching about $50 \text{ m}^3/\text{min}$.
- (b) The ISL has been operating for about 25 years, and so far 32 ISL fields have been developed, covering a total surface of about $6\,000\,000 \text{ m}^2$.

3.3. Negative impact on the environment

The quantity of 3.8 mil. tonnes of H_2SO_4 , 270 000 tonnes of HNO_3 and 103 000 tonnes of NH_4 were injected into the leaching fields of Stráž deposit in the last 25 years. They affected a total of 188 mil m^3 of Cenomanian water in the area of 28 km^2 .

Acid solutions from ISL fields dispersed horizontally and vertically not only to the Cenomanian horizon but along the extraction boreholes and tectonic lines into the Turonian aquifer of drinking water. To solve the negative influence of coexistence of the Hamr mine and the Stráž ISL mine, a protective hydraulic barrier has been built between these two deposits.

According to hydrogeological investigation and hydrological modelling of water and acid solutions the first steps have been already executed. Circulation water from Hamr mine is pumped out in a quantity of about 50 m^3/min and forced into the Stráž hydraulic barrier. This will make a steady overpressure in the SW part of the Stráž block behind the hydrological barrier.

To solve the hydrogeological situation and the negative impact of ISL on the environment, it was decided to build a desalinization plant with a capacity of 5 m^3/min . This plant is being built just now.

4. Environmental remediation programme

As already mentioned before the environmental remediation programme is based on:

Government Decision No. 366/1992 — Complex evaluation of chemical mining in North Bohemian area. A programme for closing of ISL and remediation of deposits.

Government Decision No. 429/1993 — Concept of the recession of underground Hamr mine and its "dry" conservation.

The Government Commission of Experts, established to control and supervise the above mentioned Decisions. This commission recommended the following steps of a remediation programme:

- (1) Stop the exploitation of Hamr mine and uranium mill;
- (2) Prepare a project of membrane processes to decontaminate the turonian aquifer and to clean tailings of the mill;
- (3) Keep running a protective hydraulic barrier between Hamr and ISL fields of Stráž;
- (4) Prepare a project and start the construction of desalinization plant;
- (5) Monitoring the quality of Turonian drinking water;
- (6) Preparation of technologies for the final separation and use of aluminium-ammonium sulfate.

An analysis of problems connected with the evaluation of the coexistence of the Stráž ISL mine, Hamr Mine and the environment is being executed to complete the above mentioned remediation programme. The main contractor of this project is company DIAMO s.p. together with the cooperating institutions (e.g. Aquatest, Charles University, MEGA, Geofyzika Ltd.) and selected foreign companies. This analysis is supposed to gather all the data needed from the designed Environmental remediation programme that will be unique in its scale and time extend in the whole Europe.

According to Government Decision Nos. 366/92 and 429193 it is necessary to answer basic questions regarding ISL in this region including the statement of s.c. ecological way of ISL mining process in this area.

Once in its final form, this analysis will be submitted to the Government of the Czech Republic on 30 June 1995 and will cover even the evaluation of suggested remediation steps according to EIA.

5. COMPLEX FINAL ANALYSIS OF PROBLEMS

5.1. Geological environment and underground physical and chemical processes

The following topics have to be solved:

- supplement and precise the conditions of geological, hydrogeological and geophysical research regarding the mine Stráž p. Ralskem including its wider environment,
- evaluate the contamination of underground water influenced by ISL,
- assess the long term hazards for the North Bohemian cretaceous strata arising from the existence of big volumes of contaminated groundwater,
- prepare hydraulic and hydrodynamic models for the establishing the suitable "algorithm" for the quantification of long term hazards,
- propose the quality limits of groundwater after finishing of the restoration of the area,
- propose an optimal regime of ISL, underground mine and remediation programme of geological environment including the quantification of future uranium production from existing leaching fields.

Objectives 1994

Geological, hydrogeological and tectonical results are in a form of digital maps prepared for GIS of District Council of Česká Lípa. This research has contributed to the maximum understanding of geological and tectonical development of the area. By means of hydraulic modelling the planning of potential endangered important Cretaceous aquifers has been designed.

The final report will be ready for discussion and review at the end of 1994.

5.2. Technologies applied for the remediation of the environment

Thermal thickening by evaporation followed by dressing of concentrated components was chosen for the decontamination of strongly salinated waters. The expected pumping volume of 5 m³/min will have to be extracted from the Cenomanian aquifer. For the decontamination of slightly polluted water from the Cenomanian and Turonian aquifer the volume of about 2-3 m³/min. has to be filtrated by using the membrane method.

Decontamination of polluted waters can be divided in two basic phases:

- (a) Thermal thickening in evaporation station with the separation of precipitated salts from the concentrated solution. This process will be followed by dressing of separated salts focussed on the commercial application of final products.

The amount of chemicals injected to the leaching fields 1968 - 1991

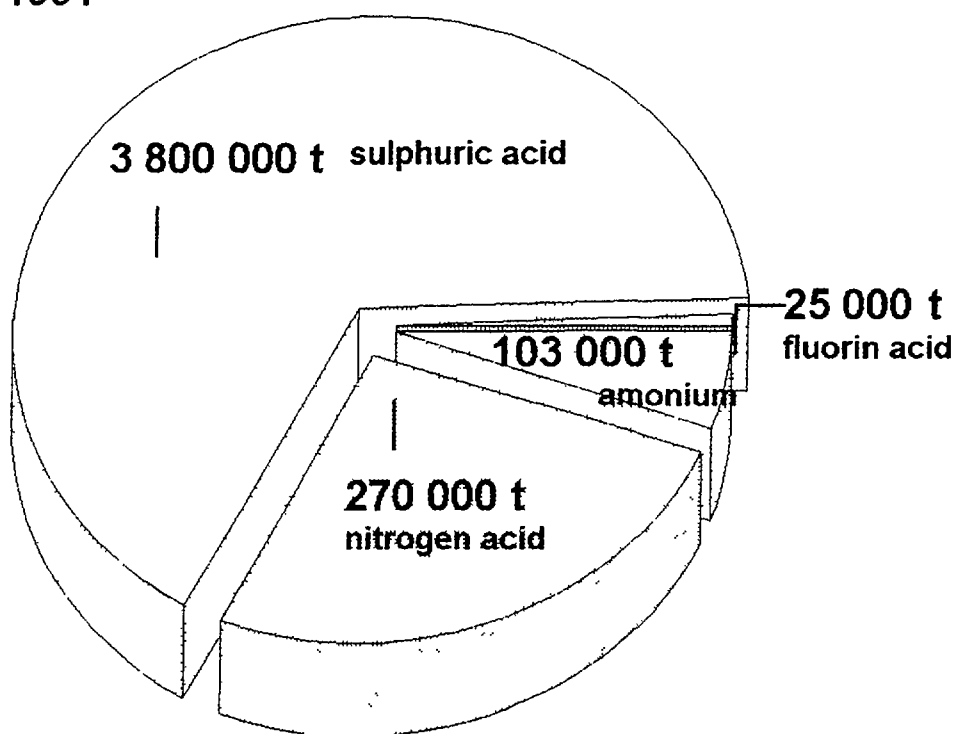


FIG. 4. The amount of chemicals injected to the leaching fields in Stráž pod Ralskem 1968–1991.

- (b) The building of complex unit for the separation of components produced by the evaporation station (Fig. 5). The station is principally based on multigrade crystallization of components and on their final dressing to an industry usable form. This is namely aluminium-ammonium sulphate which will be transferred to aluminium oxide. The sulphate constituent of aluminium-ammonium sulphate will be transferred back to a form of sulfuric acid. The remaining constituents are supposed to be transferred by means of calcination to insoluble waste suitable for the storage.

The present estimation of time needed for the decontamination of the whole Stráž pod Ralskem uranium deposit and the Hamr Mine lies between 50 and 70 years.

Objectives 1994

As a contractor of the first stage of decontamination (vaporization station) The Resources Conservation Company (RCC), USA have been chosen by the governmental commission from the public marketing tender in March 1994. This firm will build an vaporization station within the complex of former uranium mill using the old housing facilities. There have been stated technological capacities of this station that have to be met according to the signed agreement. The station will start operation in March 1996.

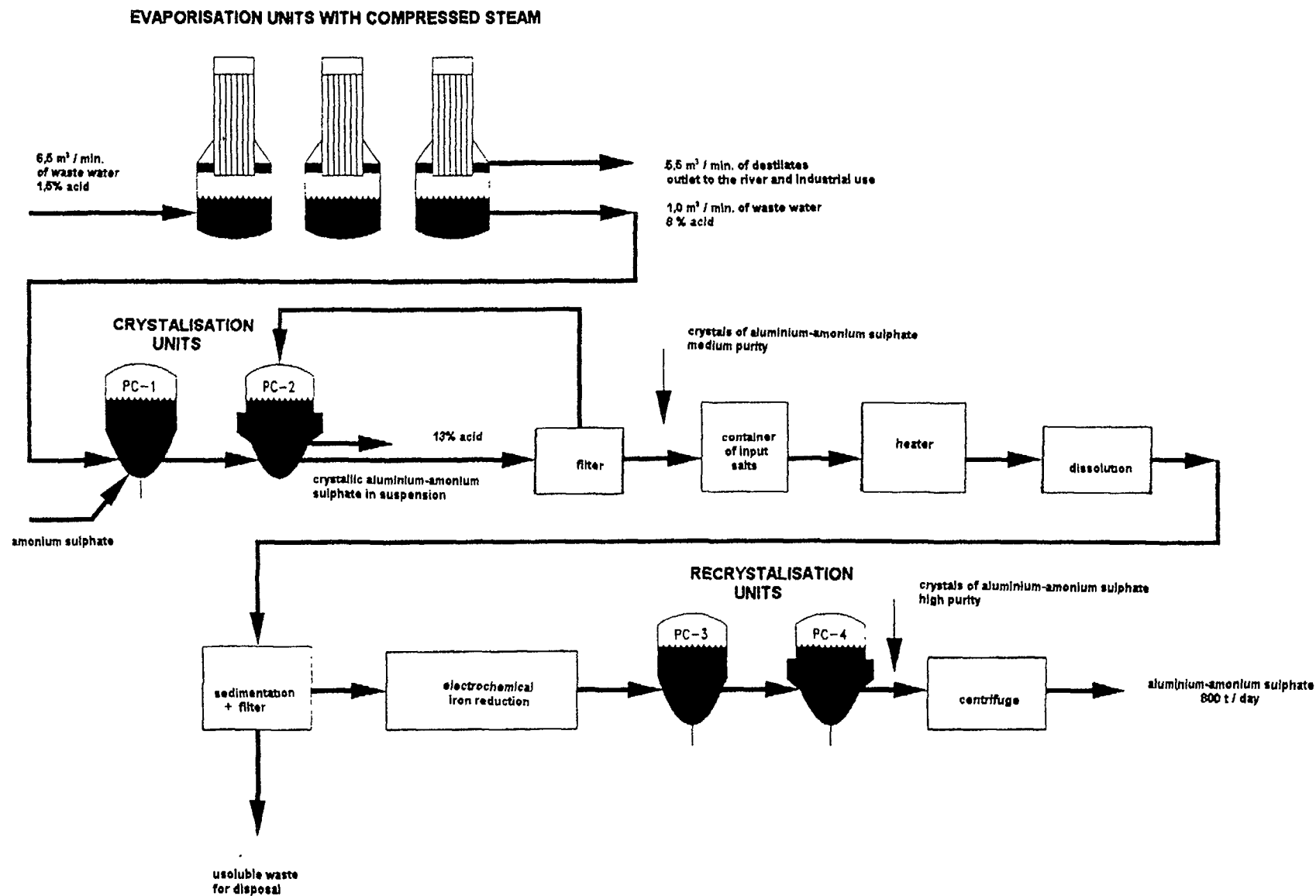


FIG. 5. Simplified diagram of vaporization station.

In 1995 the marketing tender for filtration processes will be held and chosen contractor has to start to run the decontamination of slightly polluted waters from turonian aquifer and from the tailings. The timing of this operation will be connected with the start of vaporization station.

5.3. Complex ecological assessment of remediation programme

This part of the analysis will deal with The Environmental Impact Assessment (EIA) . It is supposed to evaluate not only the negative impact of mining on the environment and to define the ecological limits of coexistence between the slowing down process of ISL and proposed remediation programmes stated above.

It is supposed to be completed by MEGA Comp. by 30 June 1995 while evaluating all remediation steps executed by DIAMO State Enterprise together with co-operating institutions according to Government Decision No. 366/1992.

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INDUCTION LOG: A GOOD WATCHDOG

Induction well logging through the PE casing for acidification measurements of groundwater and rock environment in the ISL area of the Stráž uranium deposit in the North Bohemian Cretaceous, Czech Republic

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Abstract

The Stráž uranium deposit in Northern Bohemia is exploited using the ISL method. For this method, it is necessary to observe spatial extension and time changes of sulfuric acid leaching solutions. Hydrochemical sampling of groundwater ensures this service during the whole production period. If it is possible, it is advisable to complete these analysis with the well logging results for more detailed spatial zoning. The leaching solution, which is almost completely made up by sulfuric acid, significantly changes the electric conductivity of groundwater and consequently of water-bearing rocks. The surface geophysical measurements cannot usually be used. The majority of injection, recovery and hydrogeological wells is cased by polyethylene casing. This circumstance does not allow to use the classical well logging methods for measurement of electric resistivity, such as laterolog. But it allows to use the induction log. A suitable small tool AK 2 for the high frequency induction logging was designed in 1977 (Prof. Pralat, Poland). Several of these tools have been used for the periodic checking of hundreds of wells in the ISL plant since 1979. This paper briefly recapitulates the methodology and brings typical examples of results of this method. It also informs of important innovations in this method.

1. CASE HISTORY

The site of the mining district is underlain by Cretaceous sediments with thicknesses ranging from 150 to 300 m. There are two aquifers — the upper aquifer located in Turonian and the lower in Cenomanian rocks. The two aquifers are separated by a 70 to 80 m thick sequence of siltstone and marlstone which acts effectively as an aquiclude. The uranium ores are found in the lower part of the Cenomanian sandstone sequence (Fig. 1). Two different methods are used to mine the uranium: (1) deep mining by means of a shaft and drifts and (2) solution mining using a leaching solution made up primarily of sulfuric acid in concentration from 30 to 50 g/l [1, 6]. A dense network of polyethylene injection and recovery wells screened in the ore zone is used to recover the ore. Our interest is focused on the ISL plant and its immediate surroundings.

Initial electric properties of rocks are these: (1) For the Turonian sandstones, the resistivity is approximately 500–1000 Ωm and the resistivity of groundwater is about 50–80 Ωm (hardness about 100 mg/l). The Turonian aquiclude is characterized by a formation resistivity of about 10 to 20 Ωm . (2) The original resistivity of the Cenomanian sandstones ranged between 250 and 500 Ωm with the groundwater resistivity about 30–40 Ωm (hardness about 200 mg/l). The level of artesian Cenomanian aquifer is below the free level of Turonian groundwater.

The sulfuric acid dissolved in the groundwater with low concentration is absolutely dissociated at the acid cations and sulphate anions. After years in sandstones with the clay and limy ingredient the situation changes. The acid cations are in the solution substituted by various cations, mainly aluminium. Acid solution is continually refreshed in the good permeable rocks around ore zone. A part of leaching solution escapes to the upper zone of Cenomanian aquifer or beyond borders of the ISL plant, or somewhere after an accident in the drinking water of Turonian aquifer. In this case the acid solution is not refreshed and its acidity decreases.

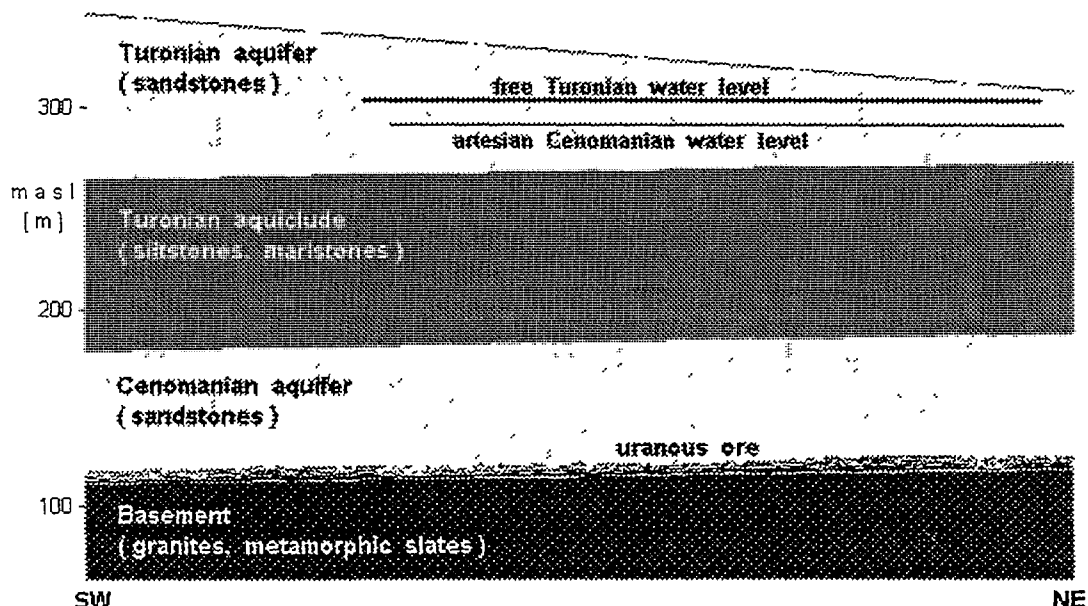


FIG.1. Sketch of geological cross-section through mining district.

The chemical analysis of pumped liquid gives an important part of information about pore liquid. But these results may be influenced by location of the well screen. Generally, this observation cannot give any information about vertical zoning of contaminants. This is because in the well screen are all screened contaminated zones mixed in one representative sample.

The presence of leaching solution in groundwater increases the concentration of electrolyte. The first consequence of this fact is a bigger electric conductivity of pore liquid, the second one is a bigger electric conductivity of all the permeable rock.

Geophysical surface measurements are applicable only for detection of massive shallow contamination in Turonian aquifer. The low reliability has two main reasons: the big electric conductivity of the Turonian aquiclude on the background and many metal objects on surface.

Well logging may be very good in this situation. From the electric conductivity of rocks and hence of electrolyte it is not possible to determine the chemical ingredients, but well logging gives a detailed information about vertical zoning of the contaminant around the well.

The question is, how to measure electric conductivity of surrounding rocks. In the new boreholes, which are not cased with pipes, there are many possibilities by using a variety of classic resistivity methods. But using of these methods is terminated the moment the well is cased. Only if the pipes are non-conductive, the induction logging can continue.

For the interpretation of pollutants from well logging it is necessary to know the relationship between electric conductivity of pore liquid and electric conductivity of all permeable rock. Very detailed studies of this relationship were made for the Cenomanian sandstones in northern Bohemia [2]. The formation factor defined as the ratio of both conductivities is constant for constant porosity and for the conductivity of pore liquid bigger than 2 S/m. For smaller conductivity of pore liquid the ratio is not constant, because there are some clay and silty ingredients in the sandstones. For all Cenomanian sandstones are the dispersion of porosity and the dispersion of volume of clay or silty fraction sufficiently low, so that the average function could not be determined.

The chemical interpretation of electric conductivity of contaminated sandstones is similar to the interpretation of nature gamma ray. The measured total gamma ray comes mainly from isotope potassium 40 and in the uranium ore zone from isotope bismuth 214, but mostly the equivalent concentration of uranium in natural uranium sequence is calculated. A similar way is used, when all chemicals in electrolyte are interpreted as one electric equivalent concentration of sulfuric acid.

By using electric equivalent concentration of sulfuric acid for interpretation of well logging, it is always necessary to consider, which chemicals are in groundwater. The same concentration of various chemicals causes various electric conductivity of solution. This is because the same mass of ions in the electrolyte is represented by a low number of the bigger slow ions or by a high number of the smaller quick ions (Fig. 2).

The relationship between electric conductivity of Cenomanian sandstones and concentration of sulfuric acid in the pore water was determined mainly for use in the sandstones around ore zone, but is used for other similar sandstones too (Fig. 3).

The relationship between the interpreted value of concentration of sulfuric acid and the mean deviation of this value shows, where the limits of estimation are. This relationship contains an inaccuracy of induction logging with the tools AK 2, the variability of porosity and the clay ingredient and finally a small influence of low dissociation of sulfuric acid by higher concentration. The lower limit was determined by the value 0.5 g/l. For higher values of concentration reliability of interpretation is acceptable. The smaller values ranged between 0.3–0.5 g/l are often detectable too, but with very low reliability.

Various properties of used induction logs are very important for the reliability of interpretation. For the described purpose the high frequency induction log AK 2 is excellent [5]. This tool can measure values of electric conductivity about 5 mS/m to 3 S/m.

The best feature of AK 2 is its radial characteristic of spatial sensitivity. For measurement of electric conductivity in a well it is always necessary to know, which part of measured value comes from surrounding rocks and which part is from the well. The interest is focused on the rocks and that is why the influence of the well must be minimized. The look on radial characteristic of spatial sensitivity shows that the contribution from the well is negligible.

According to producers' information the maximum sensitivity is about 0.5 m (from the axis), and the maximum range about 2 m. The measurement is always the same, whether there is air, fresh water or acid solution in the well, or if the diameter of borehole is 20 or 50 cm. But the influence of injected liquid in the near zone around the borehole can be well detected (Fig. 3).

The type of casing influences many properties of injection and recovery wells. Wells of the first generation were made with simple polyethylene casing and cement bond between the pipe and the wall of the borehole. The induction logging is possible at all depth intervals of this well. It can check the leaching solution or other contaminants in both aquifers. The secondary steel casing over Turonian aquifer in second generation wells protects this aquifer better when the borehole is drilled, and later when the well is used. The disadvantage of this steel casing is the impossibility to measure induction logging (Fig. 4).

The leaching fields with wells of the first generation make up about 40% of the ISL plant. Results of induction logging in Turonian aquifer agree well with hydrochemical testing of this aquifer. Hydrochemical testing of Turonian aquifer on the leaching fields with wells of the second generation show predominantly clean fresh water at the present time. For the future there are projects for several shallow Turonian wells with polyethylene casing for induction logging.

The good spatial zoning of leaching solution from induction logging may be illustrated in the detailed view on the ore zone. The uraniferous ore creates in the sandstone the fine-grained ingredient

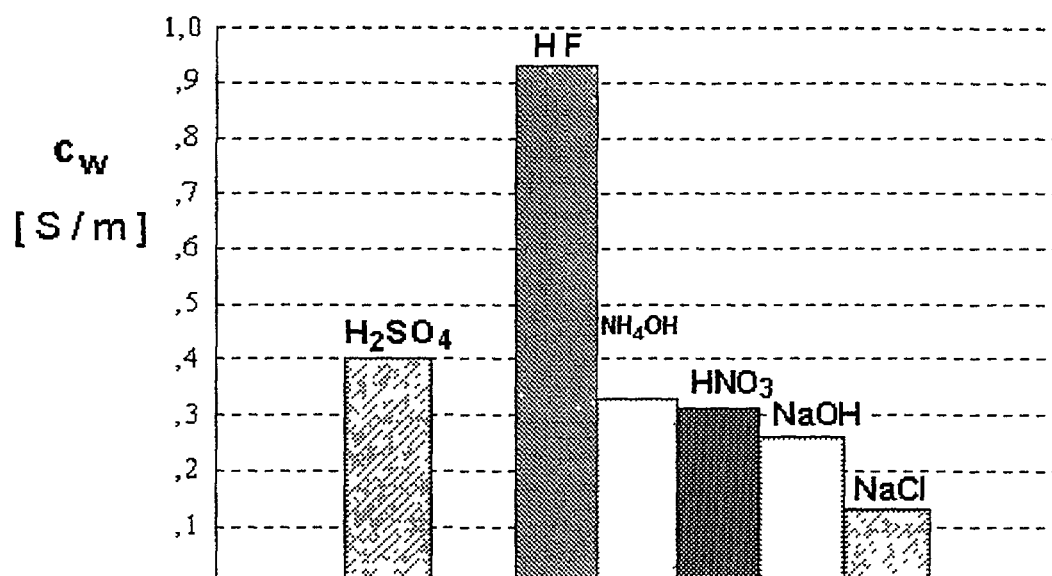


FIG. 2. The electrical conductivity of various chemical solutions with concentration 1g/l.

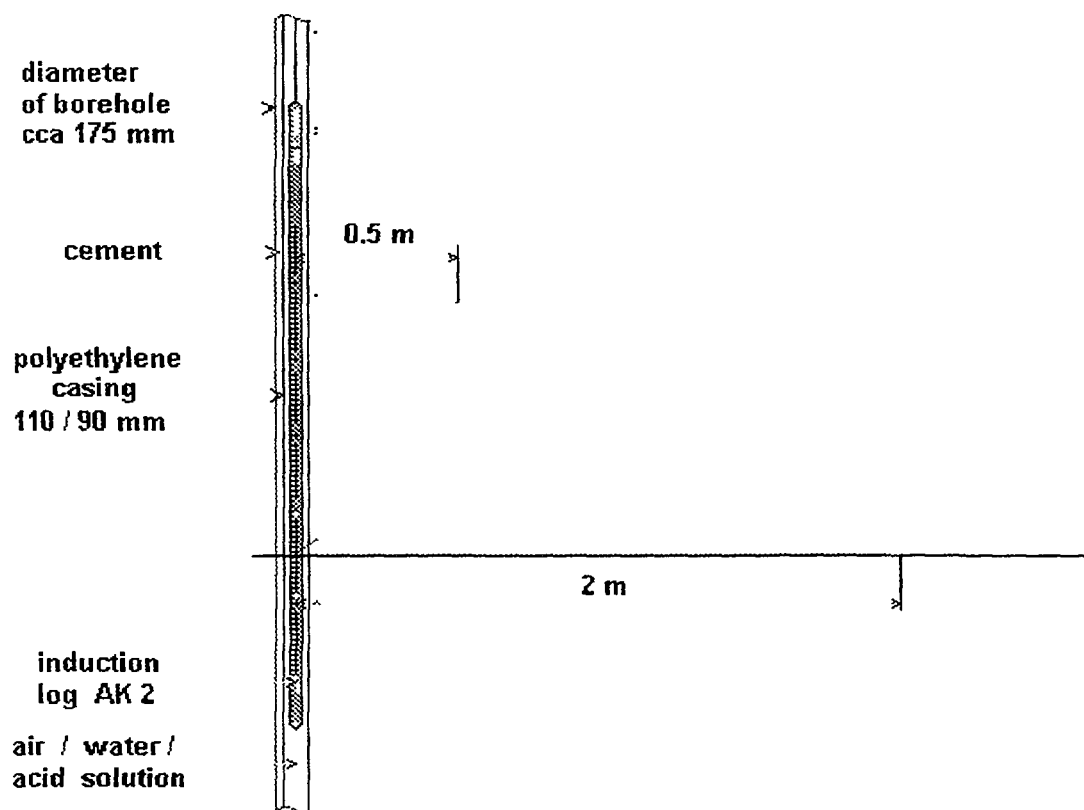


FIG. 3. The radial characteristics of spatial sensitivity of the induction log AK 2.

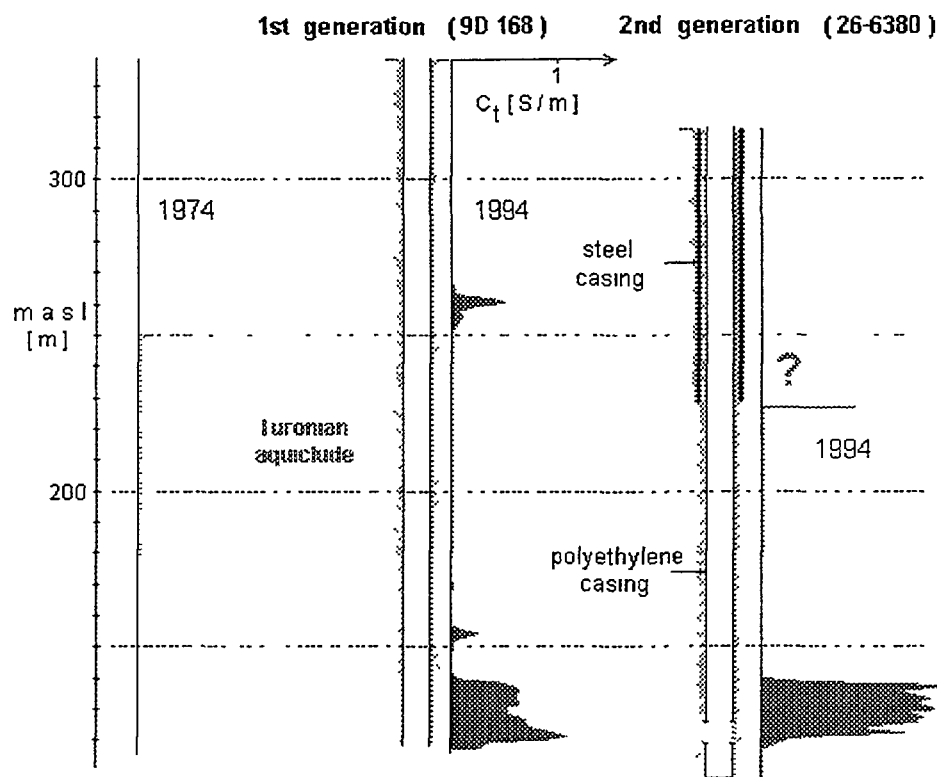


FIG. 4. The construction of first and second generation wells.

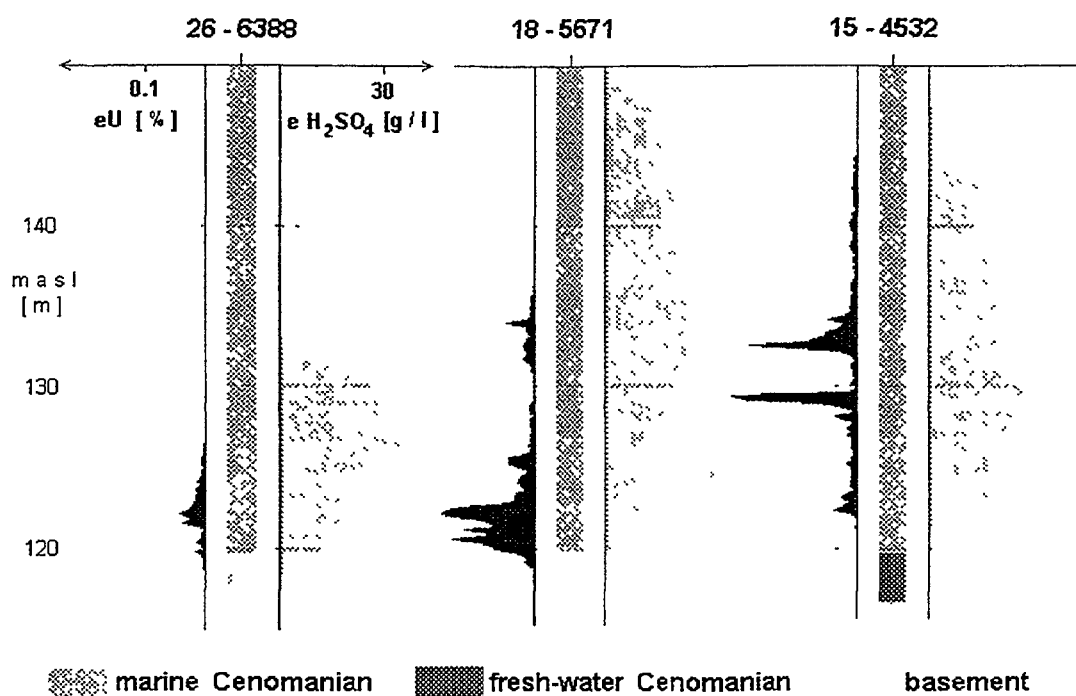


FIG. 5. The leaching solution around the ore zone.

and hence decreases permeability of the rock. The zones with maximum concentrated ore are mostly leaching slowly. This is an absolutely typical phenomenon, but after several years of leaching the ore is recovered sufficiently (Fig. 5) [4].

The detailed look on the clean and contaminated sandstones allows other speculations. In general, if the electric conductivity of the rock increases, the equivalent concentration of pore electrolyte increases too. Another relationship appears, when we focused in the long term contaminated layer of sandstones with anticipated constant concentration of pore electrolyte. The recorded variability of electric conductivity of leached rock is caused mainly by the variability of porosity here.

There are two ways to describe the presence of contaminant in permeable rock: the content of contaminant per unit volume of rock — the quantity Q_{as} and the concentration of contaminant in pore water — the quantity Q_{asp} . The relationship between these two quantities is:

$$Q_{asp} = Q_{as} / p \quad (1)$$

where p means effective porosity. The earlier determined relationship between formation factor and porosity [2] leads us to the conclusion, that the electric conductivity of sandstone has a better correlation with Q_{as} than with Q_{asp} . This is the reason why the more exact interpretation must be:

$$Q_{as} = f(c_t) \quad (2)$$

where c_t is the total electric conductivity of rock. The quantity Q_{asp} may be just an estimate from more exact Q_{as} by using average value of effective porosity. For some purposes it is necessary to make this estimate too.

It is good to know, that the quantity Q_{as} is interpreted more exactly, because this quantity is usable for other important estimates. The first estimated parameter is areal density of electric equivalent sulfuric acid Q_a [kg/m²]. It is the total mass per unit area. This parameter integrates the quantity Q_{as} over fixed depth interval.

Measurements in wells close to each other are very similar. This fact allows to anticipate a sufficiently inerratic array of parameter Q_a . Hence we can use this parameter to create contour maps of contaminant in a fixed layer. Finally the parameter Q_a can be integrated over fixed area to the quantity Q . It is the total mass of electric equivalent sulfuric acid in fixed volume.

In the Turonian aquifer on the ISL area the groundwater pollution is caused by a variety of sources. The main causes are several surface accidents in the past with used chemicals and leakage from surface pipes supplying the leaching solution or leakage through cracks in polyethylene casing. The induction logging measurements indicated massive pollution in a limited area of the leaching field No. 9. As a result, the decision was made to remediate the Turonian aquifer by pumping shallow Turonian well VPCT situated near the center of contaminated area. The progress of remedy is clearly documented by a time series of induction measurements from the injection well 985 situated near the well VPCT (Fig. 6).

A time series of induction measurements were made from 27 injecting wells around the well VPCT. Temporal variations of parameter Q_a for contaminant in Turonian aquifer was determined from these wells. Finally, the total amount Q in fixed circle was calculated too. The software product SURFER allowed to visualize these 4-dimensional data joined together (Fig. 7).

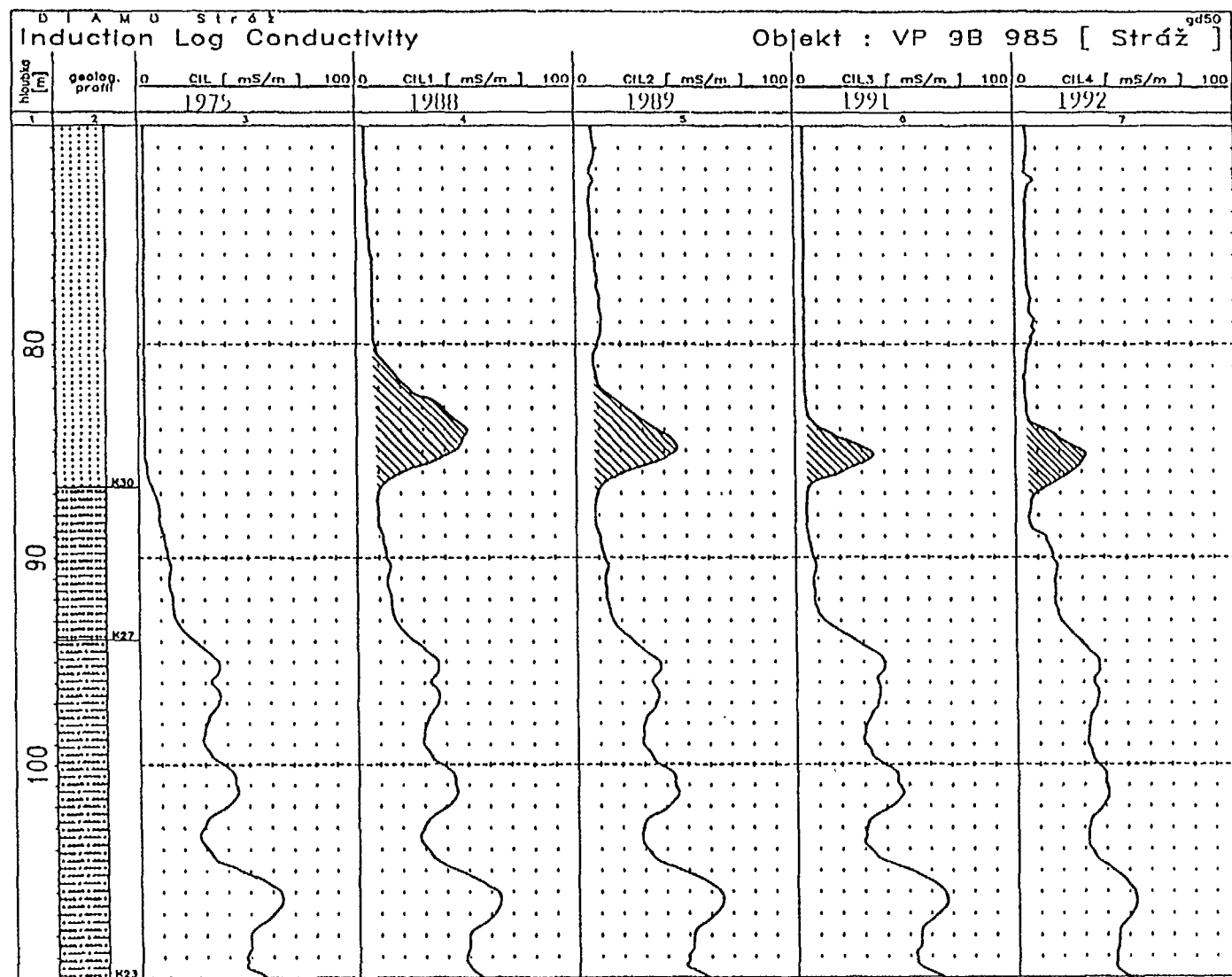


FIG. 6. The time variations of contamination in the Turonian aquifer around the well 985.

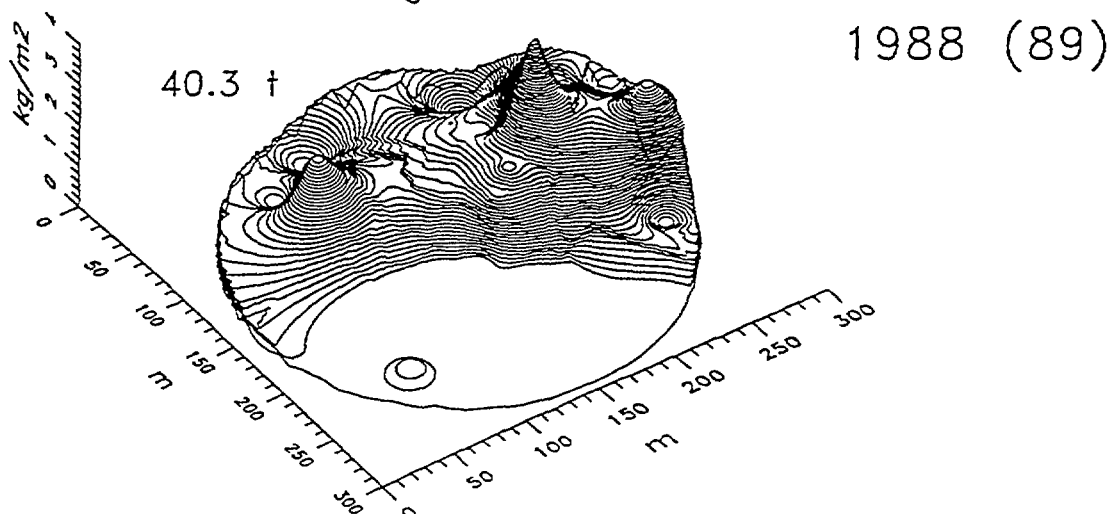
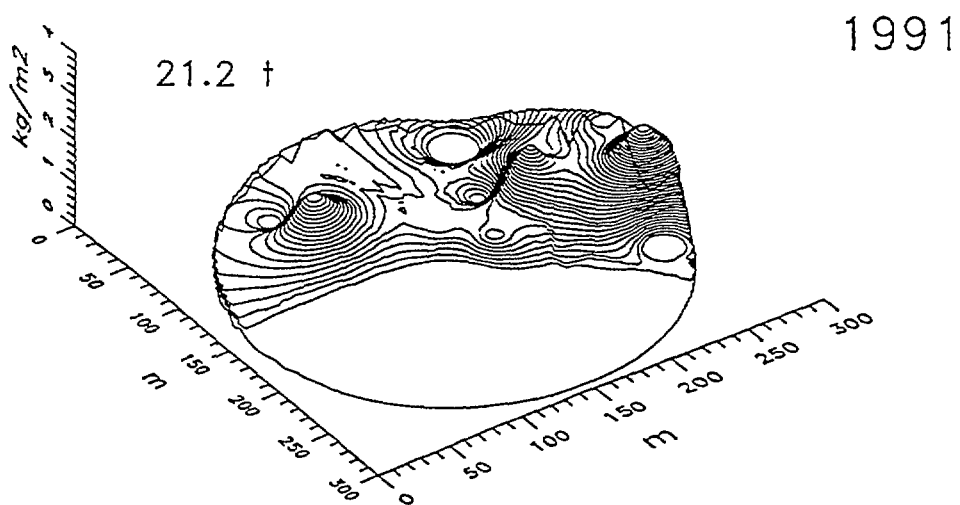
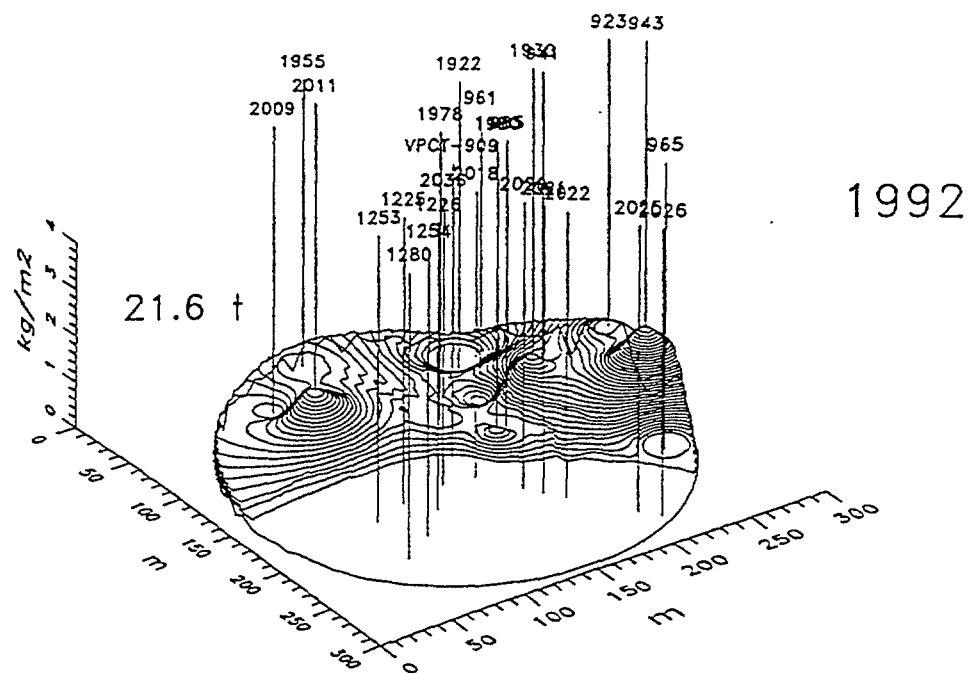


FIG. 7. Visualization of temporal and spatial variations of pollution in the Turonian aquifer.

2. INNOVATIONS

At the beginning of induction logging on the ISL plant we decided to record measurements from the field in a digital form for the next automatic interpretation. This registration was made on the paper punch tape and the automatic interpretation was made on an old large-scale mainframe computer. This way was not effective and was terminated. After that the manual interpretation was made in a simplified way for many years. But the last four years we use again automatic interpretation, which is now more effective thanks to personal computers. A well logging operator records the measurements from wells only on the hard disk of a notebook and there is no delay with graphic registration on the paper band [7].

The primary electric output from induction logs is converted into electric conductivity of rock c_i and can be converted directly to content of leaching solution Q_{as} or to estimated concentration Q_a for sandstones. The areal density Q_a for fixed depth interval is calculated after the manual determination of the position of depth limits.

The results of manual interpretation were stored in the simplified tabular form. These results were saved in a database disk file for easy use. That disk file is updated in the same way from automatic interpretation too. At present it contains approximately 11 000 records, where every record represents one interpreted depth interval with average concentration Q_{asp} together with the date.

An important innovation was made in the searching for cracks in polyethylene casing too. That search is absolutely necessary, predominantly in the wells of first generation, where the leaching solution can leak through some fissure to the Turonian aquifer. These cracks appear in about 10 to 20 wells every year. These wells must be liquidated. The group of old instruments used to search for cracks with analog graphic registration were worn out. Therefore these instruments were replaced by 6 new small hand driven apparatuses made by our department in the past two years.

The new small apparatus records well logging measurements in digital form in the memory of organizer PSION. These measurements are transferred to the PC, where the updated database is located. Thanks that fact, reliability of interpretation of measurements increases, because the interpreter can easily compare the new measurement with the older ones directly on the display. Every year about 5000 to 6000 of these measurements were made.

3. CONCLUSION

In the last years, mining in the described district was partially restricted. In the future we can expect a transfer of interest from mining to remediation. Geological observation should continue despite expected poor conditions. For the interpretation of induction logging the final step will be increasingly important, where the electric equivalent concentration Q_{as} will be substituted by the real concentrations of the chemicals present in groundwater.

That is why there are twin projects of Turonian wells for the next year. For a more complex study of Turonian aquifer it is necessary to make the time series of induction logging and the time series of chemical testing in the important places. The location of well screen in the well for pumping and chemical testing will be determined according to results of well logging and core. The number of twin projects will depend on the economic situation of our company. There will probably be about 5 twin projects for 1995. The results could help to colour our black-and-white film.

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THE CROW BUTTE ISL PROJECT: A CASE HISTORY

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Abstract

The Crow Butte project, located in the northwest corner of Nebraska, is one of four commercial in-situ uranium leach mines currently operating in the United States. The facility began commercial production in April 1991 some 12 years after the discovery of the sandstone, roll front uranium deposit. The ore body lies about 700 feet (213 metres) below the surface in a highly permeable aquifer with excellent shale/clay confinement above and below. The exploration and development period included ore body delineation, pilot testing with successful demonstration of aquifer restoration, feasibility study, environmental licensing, engineering design, and facilities construction. The commercial startup is taking a phased approach with current annual production of some 700 000 pounds U_3O_8 ; increasing to a planned one million pounds by 1997. Operations during the four years of commercial production, using alkaline leach chemistry, have gone smoothly with no major technical or regulatory problems. Capital costs for construction of the 3 500 gallon per minute (221 litres per second) capacity plant were minimized by acquiring used processing equipment from closed in situ leach uranium mines located in Texas and Wyoming. The total manpower requirement of only 25 on-site employees contributes to the low production costs.

1. INTRODUCTION

An earlier paper explained the history of the project and results of the first 1 1/2 years of operations [1]. This paper updates the earlier paper and adds the results of an additional two years of operation.

The Crow Butte in-situ leach (ISL) uranium mine, located in northwest Nebraska, commenced commercial production of yellow cake in April 1991. The well field is presently operating at 3 300 gallons per minute (208 litres per second) with a plant capacity of 3 500 gallons per minute (221 litres per second). A summary of Crow Butte parameters is given in Table I. The mine uses an alkaline leach chemistry (sodium bicarbonate) with oxygen as the oxidant. The ore body, discovered in 1980 [2], is some 600–700 feet (183–213 metres) below the land surface in the present mining area. The mineralization occurs in a typical sandstone, roll front deposit. The deposit has an overall ore grade of about 0.25% U_3O_8 . Pilot scale mining that took place in the 1986–87 time period demonstrated the technical and economic feasibility of both the mining and aquifer restoration phases of the project.

The environmental licensing of the project with the state and federal regulatory authorities has been somewhat slower and more difficult in comparison with other ISL mines in the United States primarily because the Crow Butte project is the very first uranium mine in the state of Nebraska. Nebraska did not have any regulations covering uranium mining prior to the discovery of the Crow Butte deposit. In fact, there has been no mining activity for any minerals in Nebraska other than sand and gravel. It was therefore necessary for the state to research, prepare, and approve a set of ISL

TABLE I. CROW BUTTE DEPOSIT DAWSON COUNTY, NEBRASKA, U.S.A.

PROJECT SUMMARY**Type of Operation****In Situ Leach****Source Material**

Host rock type	sandstone
Geologic name/age	Chadron Fm./Oligocene
Transmissivity	2,700 gpd/ft
Hydraulic conductivity	9.1 ft/day (2.8 meters/day)
Porosity (percentage)	29%
Tonnage factor (cu ft/ton)	17
Static water head above ore zone (feet)	550

Ore Characteristics

Uranium mineral(s)	Coffinite, uraninite
Average depth (ft.)	650
Average grade (% U_3O_8)	0.25
Grade cutoff (% U_3O_8)	0.02
Average thickness (ft.)	5.0
Average grade thickness (% ft.)	1.25
Grade Thickness Cutoff (% ft.)	0.5

Well Field Characteristics

Total number of operation production wells	130
Injector to producer well ratio	1.9
Production flow for project (gpm)	3,300
Average production per well (gpm)	25
Average injection per well (gpm)	13
Average screened interval (ft.)	15
Configuration (well pattern)	5-7 spot
Well spacing (prod. to inject., feet)	70
Casing, I.D. (inches)	4.5
	injection
	production
Casing material	PVC
Type of completion	Under ream
Surface elev. (ft above sea level)	3,900
Ground water temperature ($^{\circ}F$)	70
Natural ground water quality (TDS)	1,250

Process

Oxidant (concentration, mg/l)	O_2 (200)
Leachant (concentration, mg/l)	$NaHCO_3$ (1,000)
pH	7.7
Pregnant liquor grade (ppm U_3O_8)	55
Extraction process	up flow column ion exchange
Eluate chemistry	chloride and carbonate
Precipitation chemistry	hydrogen peroxide
Equipment	Process tanks, pumps, belt filter, agitators, thickener
Type of dryer	Vacuum
Waste disposal -	Evaporation ponds, deep disposal well, land application
	solids
	mill tailings

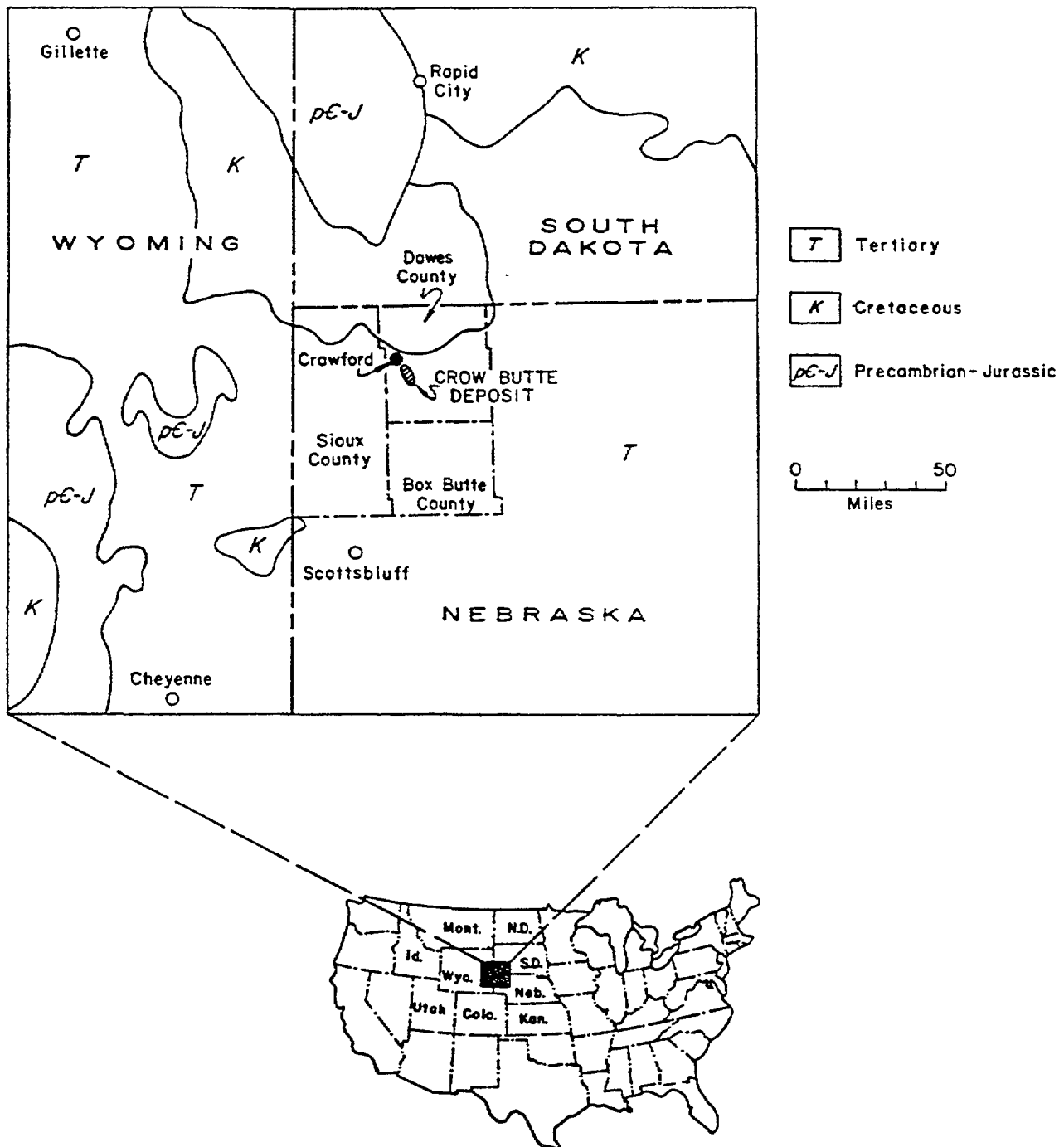
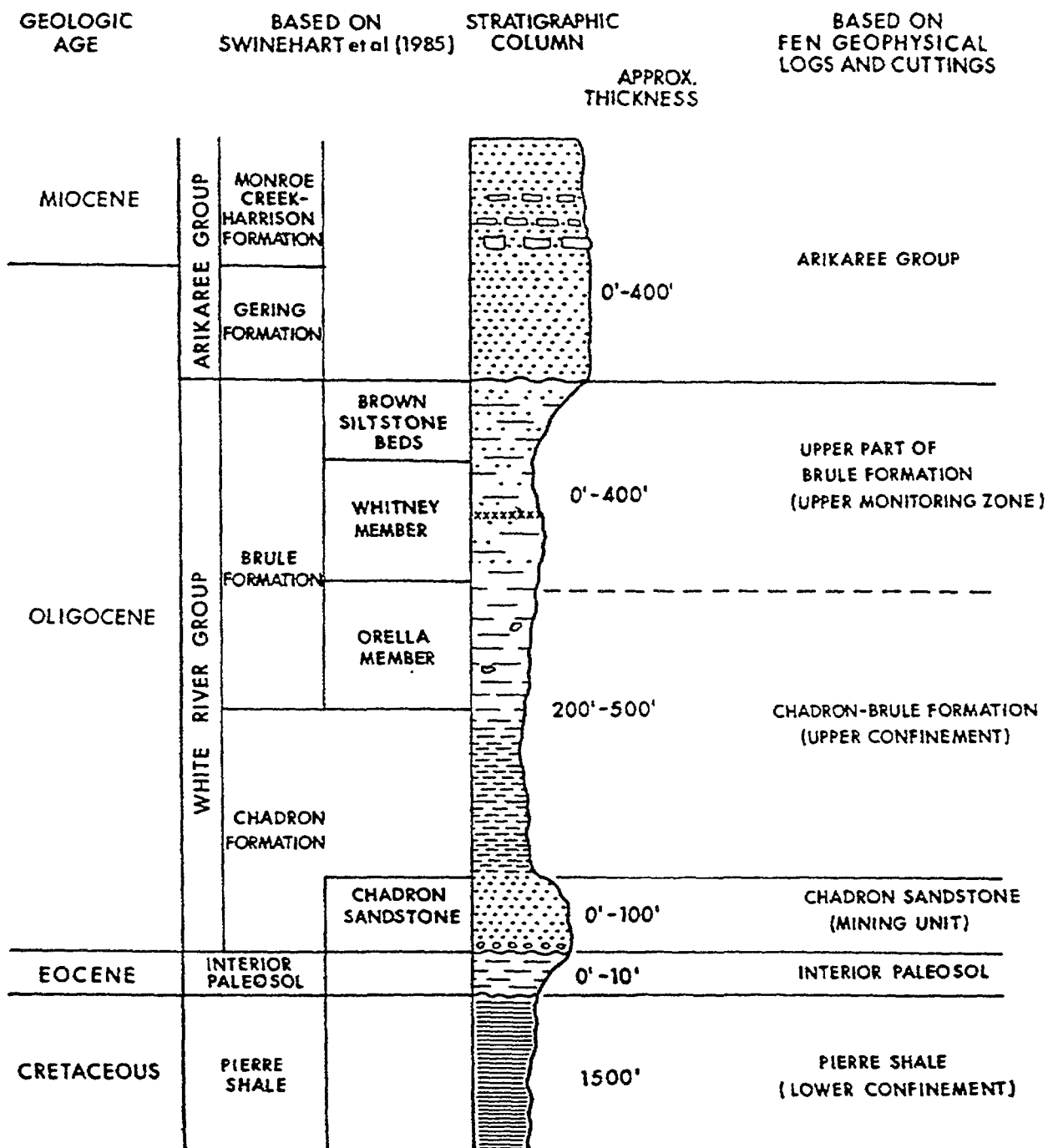


FIG. 1. Crow Butte Uranium Deposit — location and regional geology.

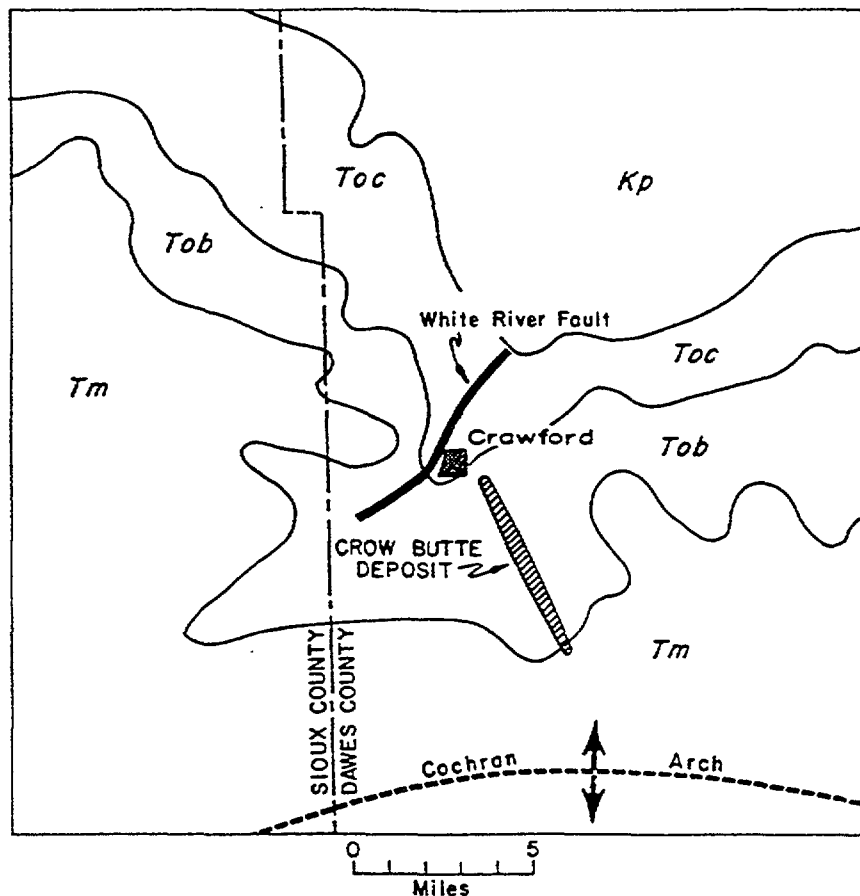
mining regulations. Now that the state officials have regulations and have become familiar with ISL mining, the regulatory process is much smoother and efficient, and license modifications are usually obtained in a reasonable length of time.

The Crow Butte project is located about 4 miles (6.5 kilometres) southeast of the small town of Crawford, Nebraska (see Fig. 1) at an elevation of 3 900 feet (1.190 metres). The rolling plains topography, with local relief of less than 100 feet (30 metres), is typical of western Nebraska and the land surface is used primarily for livestock grazing, and the growing of hay and wheat. The annual average precipitation is 15.5 inches (39.5 centimetres), and the temperature extremes are +110 degrees Fahrenheit (43 degrees Centigrade) summer and -31 degrees Fahrenheit (-35 degrees Centigrade) winter.



REV. DATE	CROW BUTTE RESOURCES, INC.	
	CROW BUTTE PROJECT	
	Dawes County, Nebraska	
	AREA OF REVIEW	
	STRATIGRAPHIC COLUMN	
	PREPARED BY: F. E. N.	
	DRAWN BY: J. C.	DATE: 2/88

FIG. 2. Crow Butte Project — Area of review, stratigraphic column.



	<div style="border: 1px solid black; padding: 2px; display: inline-block;">Tm</div>	Arikaree Group and Marsland Formation
TERTIARY	<div style="border: 1px solid black; padding: 2px; display: inline-block;">Tob</div>	Brule Formation
	<div style="border: 1px solid black; padding: 2px; display: inline-block;">Toc</div>	Chadron Formation
CRETACEOUS	<div style="border: 1px solid black; padding: 2px; display: inline-block;">Kp</div>	Pierre Shale

FIG. 3. Crow Butte Uranium Deposit, Northwest Nebraska — local geology.

The Crow Butte project is held and operated by Crow Butte Resources, Inc. (CBR) under a Joint Venture and Operating Agreement to which CBR is a party. The owner of the underlying mineral rights is the Crow Butte Land Company, a subsidiary of CBR. The beneficial owners of the project, and the owners of all production, are Uranerz U.S.A., Inc., Kepco Resources of America, Inc., and Geomex Minerals, Inc.

2. GEOLOGY

The Crow Butte uranium deposit is in the Chadron sandstone unit of the White River Group (see stratigraphic column, Fig. 2). Below the Chadron sandstone is the Cretaceous Pierre shale which is the oldest formation of interest for the Crow Butte area since it is the lower confining formation for the uranium mineralization. The Pierre shale is a widespread, dark gray to black marine shale that

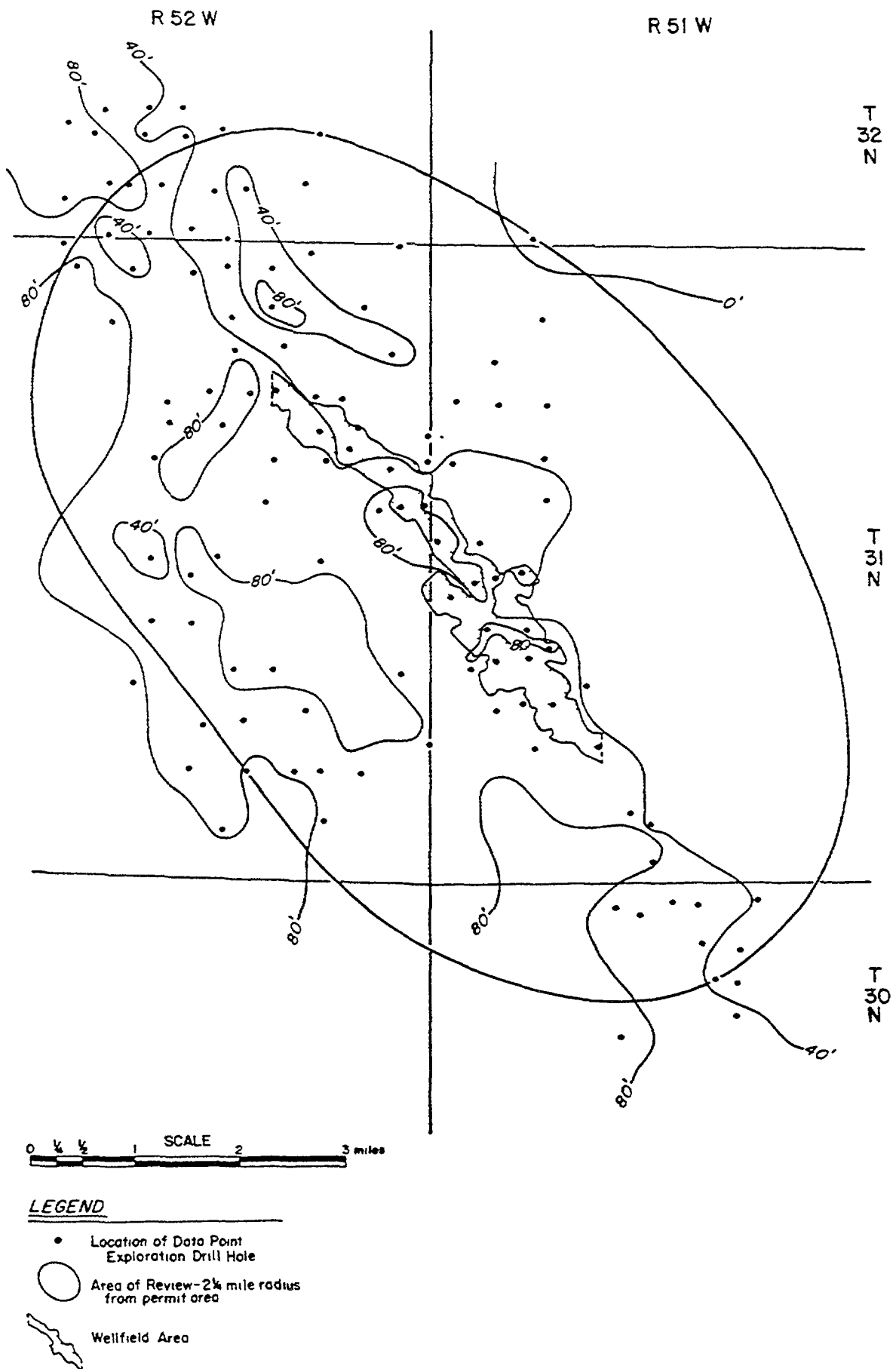


FIG. 4. Thickness-Basal Chadron.

is essentially impermeable. Aerial exposure and erosion greatly reduced the thickness of the Pierre prior to Oligocene sedimentation. The top of the Pierre is a major unconformity and exhibits considerable paleotopography [3]. An ancient soil horizon or Paleosol is locally present on the surface of the Pierre shale [4].

The White River Group is Oligocene in age and consists of the Chadron and Brule Formations (Fig. 3). The White River Group outcrops as a band at the base of the Pine Ridge in northwest Nebraska. The Chadron Formation is the oldest Tertiary Formation in northwest Nebraska.

The Chadron Sandstone is generally present at the base of the Chadron Formation and is a coarse grained arkosic sandstone with frequent interbedded thin clay beds and clay galls. Occasionally the Chadron Sandstone grades upward to fine grained sandstone containing varying amounts of interstitial clay material and persistent clay interbeds. The Chadron Sandstone is the host member and mining unit of the Crow Butte ore deposit and no other uranium mineralization is present in overlying units.

The vertical thickness of the Chadron Sandstone within the Crow Butte area averages about 60 feet (19 m). An isopach of the Chadron Sandstone in the Crow Butte area indicates a range in thickness of 0 feet on the northeast to nearly 100 feet (30 m) on the west (Fig. 4). An east-west cross-section through the wellfield area illustrates the confined nature of the Chadron Sandstone with electric logs of regional exploration holes (Fig. 5). Even greater thickness are developed further to the west. A persistent clay horizon typically brick red in color generally marks the upper limit of the Chadron Sandstone.

TABLE II. ESTIMATED WEIGHT PERCENT AS DETERMINED BY X RAY DIFFRACTION

Phase	Upper Part		
	Chadron Formation (2) (Upper Confinement)	Chadron Sandstone (4) (Mining Unit)	Pierre Shale (2) (Lower Confinement)
Quartz	22.5	75.5	26
K Feldspar	2	13	4
Plagioclase	1	9.5	1
Kaolinite-Chlorite	—	<1	9
Montmorillonite	44	<1	32
Mica-Illite	1	<1	15
Calcite	22	—	1.5
Fluorite	0.5	—	—
Amorphous	7	1	10.5
Unidentified	—	<1	1
TOTAL	100	100	100

Number in parentheses is number of core samples.

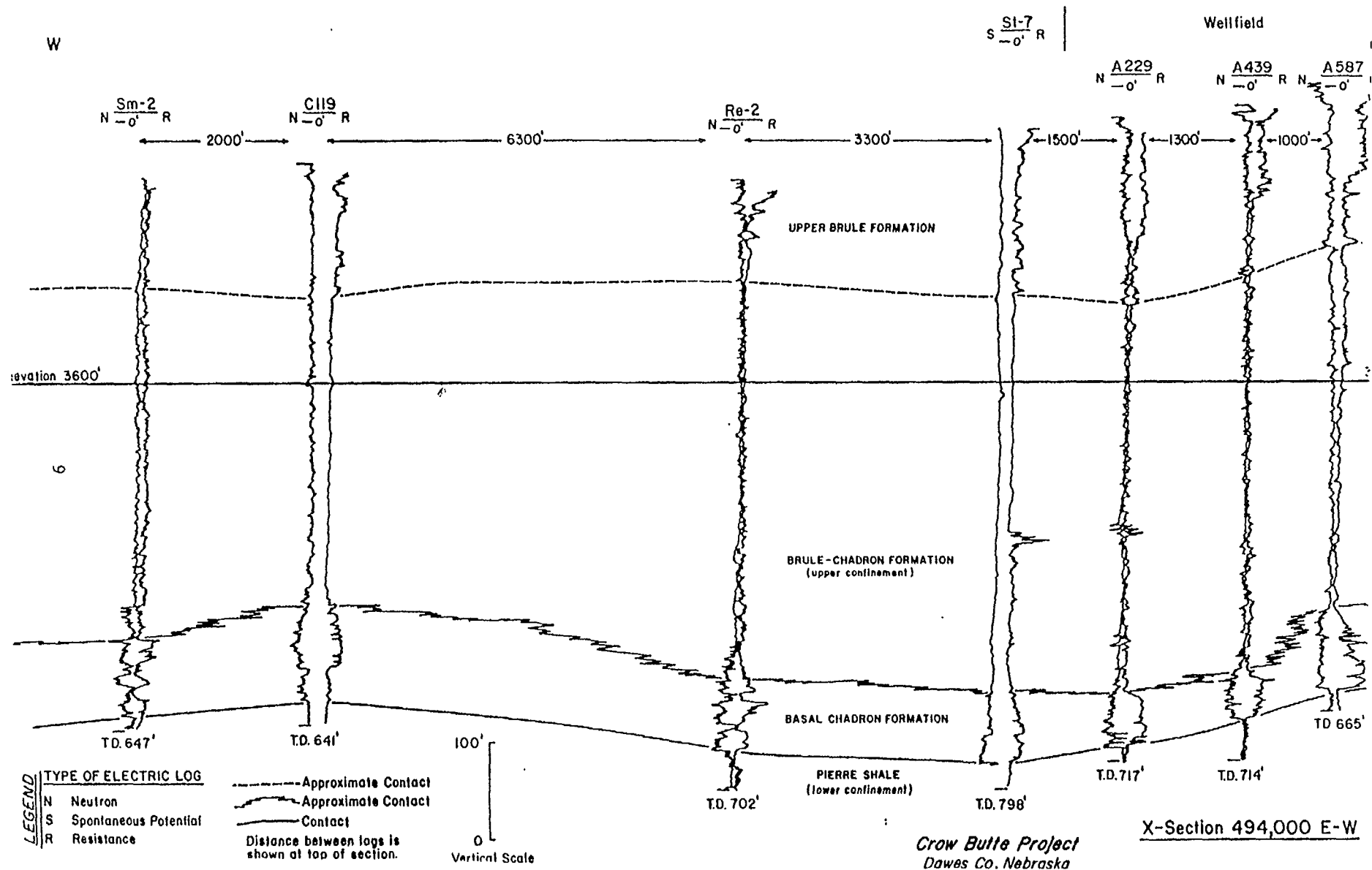


FIG. 5. Crow Butte Uranium Deposit, Northwest Nebraska — X-section 494 000 E-W.

Thin section examination of the Chadron Sandstone reveals its composition to be 50% monocrystalline quartz, 30 to 40% undifferentiated feldspar, plagioclase feldspar and microcline feldspar. The remainder includes polycrystalline quartz, chert, chalcedonic quartz, various heavy minerals and pyrite. X ray diffraction analyses indicate that the Chadron Sandstone is 75% quartz with the remainder K-feldspar and plagioclase (Table II).

Core samples and outcrops of the Chadron Sandstone exhibit numerous clay galls up to a few inches in diameter, frequent thin silt and clay lenses of varying thickness and continuity, and occasionally a sequence of upward fining sand. These probably represent flood plain or low velocity deposits which normally occur during fluvial sedimentation. Within the permit area varying thicknesses of clay beds and lenses often separate the Chadron Sandstone into fairly distinct subunits as shown on electric logs.

2.1. Chadron-Brule Formations, Upper Confinement

The upper part of the Chadron Formation and the lower part of the Brule Formation are the upper confinement overlying the Chadron Sandstone (Fig. 5). This upper confining layer is 200 to 500 feet (61 to 152 metres) thick over the Crow Butte area. The upper part of the Chadron represents a distinct and rapid facies change from the underlying sandstone unit. The upper part of the Chadron Formation is a light green-gray bentonitic clay grading downward to green and frequently red clay. X ray diffraction analyses of the red clay indicate that it is primarily comprised of montmorillonite and calcite (Table II). This portion of the Chadron often contains gray-white bentonitic clay interbeds. The light green-gray "sticky" clay of the Chadron serves as an excellent marker bed in drill cuttings and has been observed in virtually all drill holes within the Crow Butte area. The measured vertical hydraulic conductivity of the upper confinement is less than 1×10^{-10} cm/sec. The contact with the overlying Brule Formation is gradational and cannot be consistently picked accurately in drill cuttings or on electric logs (Fig. 5). Therefore, the upper part of the Chadron Formation and the lower part of the Brule Formation are combined within the Crow Butte area.

The Brule Formation lies conformably on top of the Chadron Formation. The Brule Formation is the outcropping formation throughout most of the area. The lower part of the Brule Formation consists primarily of siltstones and claystones. Infrequent fine-to-medium grained sandstone channels have been observed in the lower part of the Brule Formation. When observed, these sandstone channels have very limited lateral extent.

2.2. Upper Part of the Brule Formation — Upper Monitoring Unit

The upper part of the Brule Formation is primarily buff to brown siltstones which have a larger grain size than the lower part of the Brule Formation. Occasional sandstone units are encountered in the upper part of the Brule Formation (Fig. 5). The small sand units have limited lateral continuity and, although water bearing, do not always produce usable amounts of water. These sandstones have been included in the upper part of the Brule Formation. The lowest of these water bearing sandstones are monitored by shallow monitor wells during mining.

The structure of the Crow Butte area is illustrated on Fig. 6. Elevation contours on top of the Cretaceous Pierre Shale, base of the Tertiary Chadron Formation, illustrate the structure. The current features present in the area are a result of the erosional paleotopographic surface of the Pierre Shale prior to deposition of the Chadron Formation and some amount of structural folding and faulting which occurred after the deposition of the Chadron Formation. Regionally and within the area, the White River Group, Chadron and Brule Formations dip gently south at about 1/2 to 1 degree. The White River Fault is present along the northwest margin of the area and is dated as post-Oligocene since it cuts both the Chadron and Brule Formations. The fault has total vertical displacement of 200 to 400 feet (61–122 m) with the up thrown side on the south. The White River Fault is about one and one-half miles northwest of the northern extent of the permitted wellfield area.

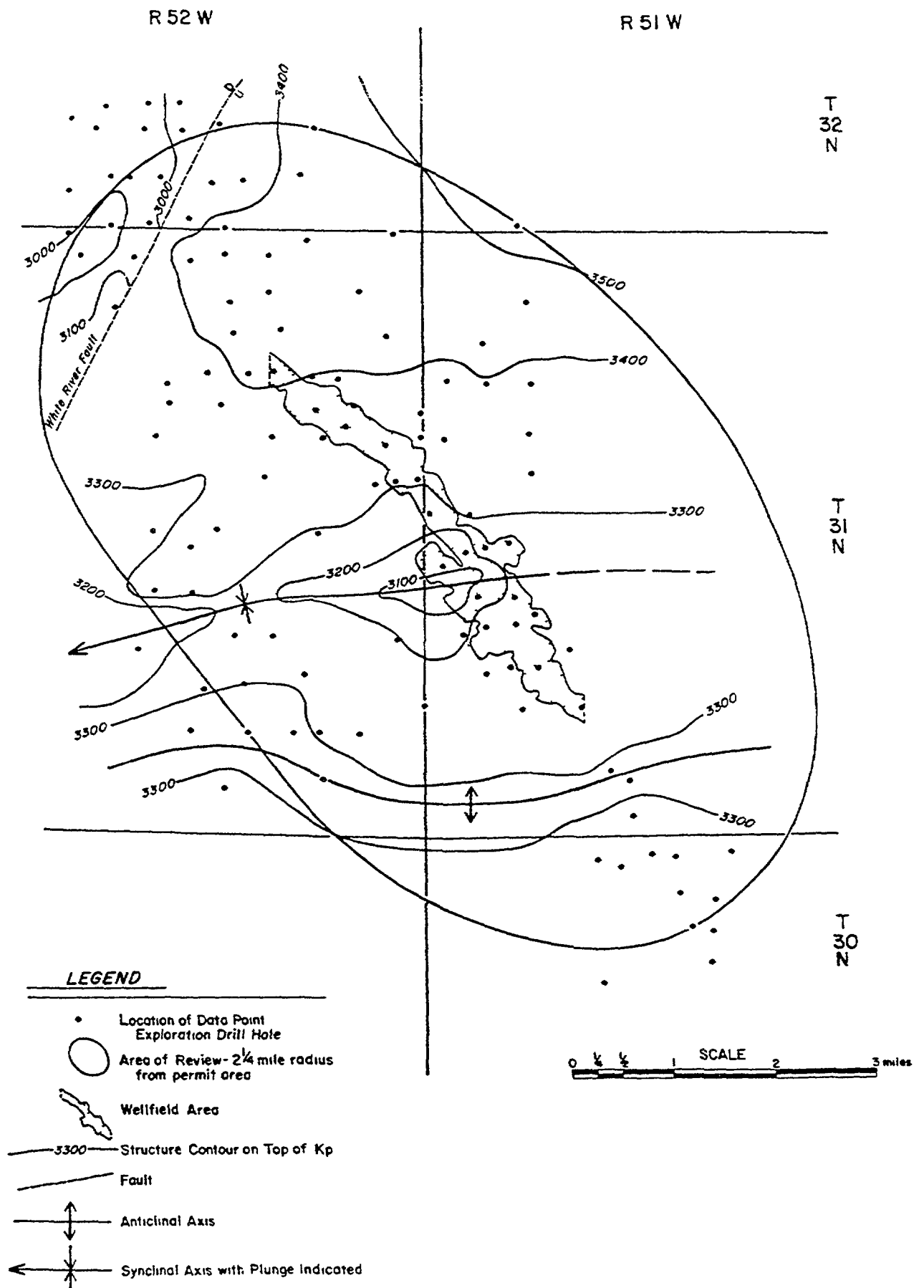


FIG. 6. Structure elevation of kp contact, Top of Pierre (Base of Chadron Formation).

Close spaced drill data throughout the area indicate that no significant faulting is present in the wellfield area. A synclinal feature trends east-west through the Crow Butte Area and plunges west. An associated east-west trending anticlinal feature is present along the southern part of the area. This anticlinal axis is subparallel to the Cochran Arch proposed by DeGraw [3] and is probably a related feature (Fig. 3).

The Miocene Arikaree Group includes three sandstone formations which form the Pine Ridge escarpment that trends from west to east across northwest Nebraska. These sandstones are fluvial in origin and are eroded in the present area of mining.

The origin of the mineralization is considered to be secondary deposition (precipitation) of uranium leached from overlying volcanic ash [5]. The dominant reductant is pyrite formed by sulfate reducing bacteria [5]. The ore body is a typical roll front deposit with the oxidation/reduction front system about 6 miles (10 kilometres) long and up to 3 000 feet (914 metres) wide. The ore, at any location, is located in from one to five horizons. The ore thickness ranges from 5 to 15 feet (1.5 to 4.5 metres) and the depth to the ore body from the surface ranges from 275 to 820 feet (84 to 250 metres). Mineralogical studies indicate that uranium mineralization is dominantly coffinite with uraninite associated with pyrite in the matrix of the Chadron sandstone [6]. The organic content of the Chadron sandstone is very low, ranging from 0.02 to 0.20% [5]. The average grade of the ore in the Crow Butte deposit is 0.25% U_3O_8 . The in-place uranium reserves for the deposit are estimated at over 30 million pounds U_3O_8 which includes mineralization outside of the present permit area.

A study of the regional geology, including the interpretation of oil and gas logs, defined an extensive fluvial sandstone system at the base of the Tertiary just above the Pierre Shale. Gamma kicks in some of the oil and gas wells, coupled with the presence of methane gas shows, provided the basis for an exploration drilling program. In 1979 a total of 95 exploration holes were drilled in the Crawford area with some weak uranium mineralization encountered. The 100th additional exploration hole drilled in 1980 intersected ore grade uranium mineralization establishing the discovery of the Crow Butte ore body. Additional drilling in 1980 defined the mineralized trend [2].

3. HYDROGEOLOGY

The basal Chadron sandstone, the uranium host sand, is water saturated and forms a regionally extensive confined aquifer that produces artesian (flowing) conditions where the local topography is depressed. The Chadron Sandstone is a coarse-grained arkosic sandstone with an average thickness in the project area of 40 feet (12 metres). The ore body aquifer, referred to locally as the Chadron aquifer, has a piezometric surface some 495 feet (151 metres) above the top of the sand at the mine dipping gently to the north-northwest. The overlying aquifer nearest the Chadron aquifer is the Brule sand (aquifer) which is separated from the Chadron aquifer by some 200 to 300 feet (61 to 152 metres) of clay, claystone and siltstone. The difference in elevation of the piezometric surface of the Chadron and Brule aquifers in the project area is about 59 feet (18 metres).

Two major aquifer pump tests were performed in the project area to establish the hydrogeologic parameters for mining and environmental purposes. The aquifer pump tests were conducted for 51 and 72 hours, and produced quality data that were used to determine the hydrologic properties of the Chadron aquifer, and the upper and lower confining layers. The results of the aquifer pump tests show that the Chadron (ore body) sandstone is a non-leaky, confined, slightly anisotropic aquifer with an average hydraulic conductivity (permeability) of about 9.10 feet per day (2.8 metres/day). The average storativity was calculated to be 1.04×10^{-4} . The pump tests demonstrated, to the satisfaction of the regulatory agencies, that the upper and lower confining units sufficiently restrict the vertical movement of water out of the Chadron aquifer [7].

4. PILOT TESTING

After extensive exploration drilling and evaluation in 1981 and 1982, the decision was made to license and operate a pilot test mine to establish both the amenability of the ore deposit to ISL mining, and the technical/economic feasibility of successfully restoring the aquifer contaminated by mining. This latter aspect was a requirement of the regulatory agencies and a prerequisite for issuance of a license for commercial mining operations. In 1982 the location for the pilot test well field was selected, and environmental baseline data were collected for preparation of the pilot test license application. The license application was submitted to the regulatory agencies in early 1983 and all necessary approvals were obtained in 1985.

Construction of the 100 gallon per minute (6.3 litres per second) pilot processing plant began in mid 1985 and was completed in early 1986.

Pilot test mining started in mid 1986 and lasted until February 1987 using an alkaline based (sodium bicarbonate) lixiviant operating at a pH in the 7.1 to 7.4 range. Oxygen was used as the oxidant and ion exchange columns using anion resin were used to recover the uranium. The restoration (recultivation) of the affected aquifer started shortly after the completion of mining and lasted until August 1987 (about a six month period). Aquifer restoration methodology included groundwater sweep, water treatment and reductant addition, and finally aquifer recirculation [7].

The pilot test results successfully demonstrated the technical and economic viability of the project in terms of both uranium extraction and aquifer restoration. The test also demonstrated, to the satisfaction of the regulatory agencies, that the mining solutions can be contained in both the horizontal and vertical dimensions. There were no excursions of mining solution during the pilot test. The project did not experience any significant regulatory or environmental problems during the pilot test, and there were no problems with regard to radiation exposure to workers.

5. COMMERCIAL OPERATIONS

5.1. Environmental Licensing

In order to construct and operate an ISL uranium mine in Nebraska It was necessary for the operator, Crow Butte Resources, Inc., to obtain project approvals from both the state and federal governments. The principal state regulatory agencies responsible for ISL mining are the Nebraska Department of Environmental Quality and the Nebraska Department of Water Resources. At the federal level, the main agency responsible for uranium mining is the U.S. Nuclear Regulatory Commission (NRC). License application documents had to be submitted separately to each of the three agencies mentioned above. Preparation of the commercial licensing applications, containing extensive information on hydrology, vegetation, soils, wildlife and radiological conditions, started in 1986 and submittal of the voluminous application documents took place in October 1987 [7].

As mentioned in the introduction section of this paper, environmental licensing was complicated and delayed by the fact that the Crow Butte project was the first uranium mine ever licensed in the state of Nebraska. The appropriate state agencies had to first write, and then get approved, regulations to cover ISL uranium mining. After much effort and considerable negotiations, the necessary environmental license and approvals were issued in April 1990. Opposition from a small but vocal environmental group lengthened the licensing by increasing the number of public meetings and hearings. The local residents are overwhelmingly in favor of the project, and the environmental group was not successful in any of its many appeals and protests.

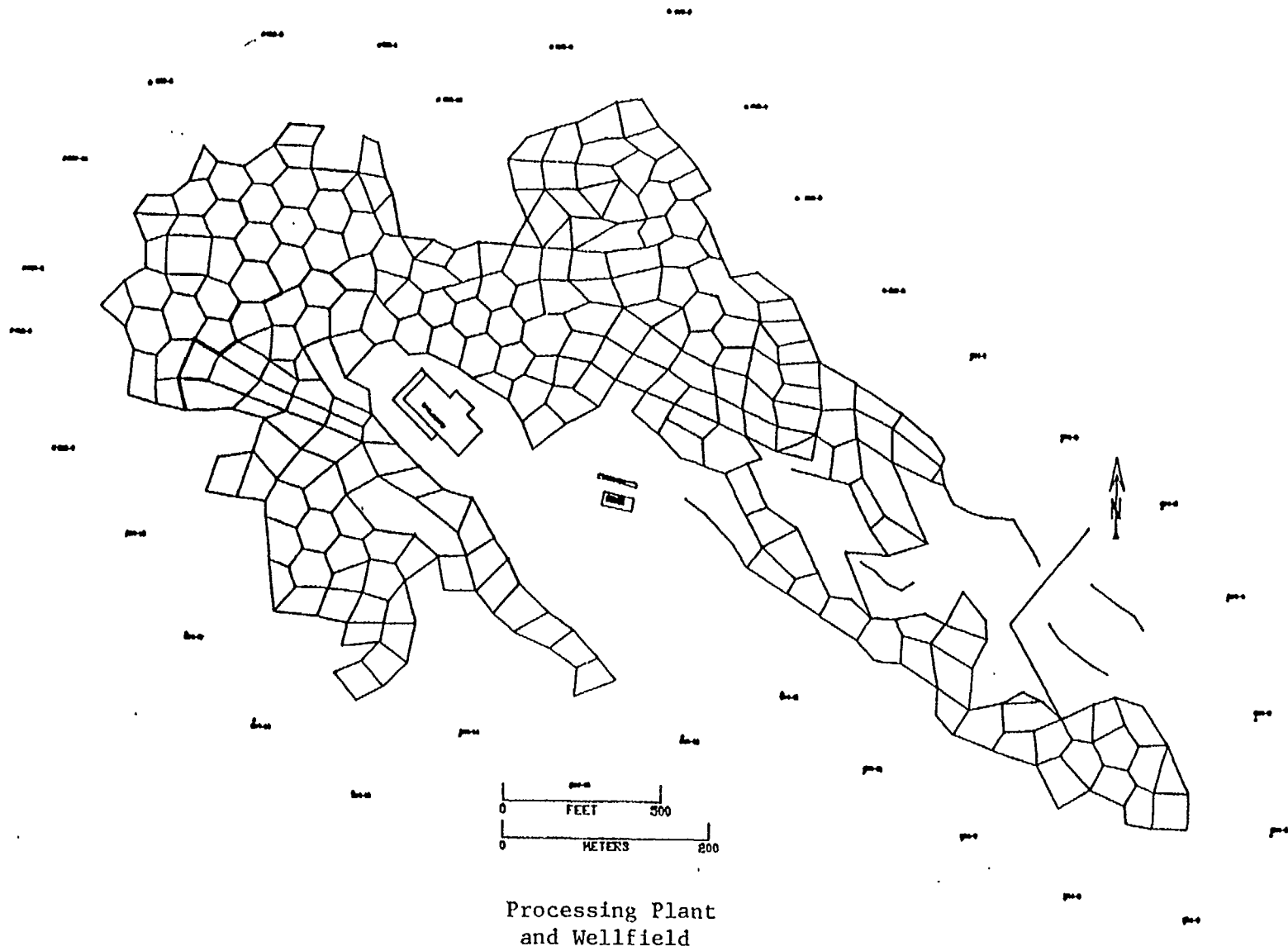


FIG. 7. Crow Butte Project, processing plant and wellfield.

5.2. Commercial Construction

The construction of the commercial ISL mining facilities started in May 1990 and was completed in March 1991. The facilities can be divided into three major components. The first component is the well field which consists of the injection and recovery (production) wells, monitor wells, downhole equipment and surface equipment. The second component is the processing plant which includes the building and the process equipment. The processing plant and wellfields are illustrated on Fig. 7. The third component is the waste water disposal system which, at Crow Butte, consists of plastic lined (double liner) evaporation ponds. Each of these components will be discussed separately.

5.2.1. Well Field Design and Construction

The wells needed to produce uranium at the Crow Butte ISL mine are installed by contract drillers using rotary drilling rigs typically found in the water well drilling business (Fig. 7). At Crow Butte three to four contract drilling rigs are employed by the operator to install injection, recovery and monitor wells. The project uses PVC Yelomine casing with 4.5 inch (11.4 centimetres) internal diameter. The well annulus is completely cemented from bottom to top, and the screen is stainless steel with a slot size of .020 inch (.051 centimetre). Well completion is accomplished using the under reaming method where only the mineralized interval is exposed by cutting away the casing and cement. Downhole pumps sized at either 5 or 7-1/2 horsepower (3.7 to 5.6 kilowatts) are hung on 2 inch (5 centimetre) PVC pipe in each recovery well. Recovery wells and injection wells are drilled and completed in an identical fashion using the same diameter casing. Following the completion of each injection, recovery and monitor well, an integrity test must be performed to ensure that there are no leaks in the casing. The results of the integrity test must be documented in writing and submitted to the regulatory agencies for review. From 15 to 20 recovery wells, with associated 3 040 injection wells, are piped to a well field manifold house using buried plastic pipe. The well field manifold house includes a flow metre, flow control valve and sample port for each well piped to the house. The electrical power on/off control for each recovery well is also located in the well field manifold house. From the well field manifold house the solutions are routed to and from the main processing plant in buried 10 inch (25.4 centimetre) high density polyethylene (HDPE) pipe. The manifold houses are skid mounted so that they can be taken to a new well field when they are no longer needed in an old well field. The project typically uses 5-spot and 7-spot patterns with a nominal distance between recovery and injection wells of about 70 feet (21 metres).

The leaching solution (or lixiviant), which causes the dissolution of the uranium, consists of Chadron formation groundwater fortified with soda ash (Na_2CO_3) and carbon dioxide (CO_2) to form a sodium bicarbonate alkaline leach chemistry. The pH of the lixiviant is kept in the 7.6 to 7.8 range. Oxygen is added to the injection solution at each well field manifold house. Oxygen is delivered to the mine and stored on site as liquid oxygen. A chemical description of the well field and process chemistry is presented in Table III [8]. The carbonate strength of the lixiviant is kept at about 1 000 mg/l CO_3 and the target oxygen concentration is 200 parts per million.

5.2.2. Processing Plant Design and Construction

The processing plant building, which also includes the laboratory, maintenance shop, warehouse and staff offices, is a standard metal building some 300 feet (91 metres) long, 115 feet (35 metres) wide and 52 feet (16 metres) tall at its highest point (Fig. 7). The major process components such as ion exchange columns, elution and precipitation tanks, product belt filter, chemical make-up tanks, pumps, piping and some electrical gear were purchased used from closed ISL uranium mines in Texas and Wyoming. The used equipment included the supporting structural steel. The purchase of used equipment resulted in substantial capital cost savings for the project. Construction of the building and process plant was completed on time and under budget, and no significant problems were encountered during the plant construction phase.

TABLE III. CROW BUTTE MINE PRODUCTION CHEMISTRY

Fortification of LixiviantBarren Lixiviant + $\text{HCO}_3^{-1} + \text{CO}_2 + \text{O}_2$ Injection WellsOre BodyUranium: $\text{UO}_2 + 1/2\text{O}_2 \rightarrow \text{UO}_3$ $\text{UO}_3 + 2\text{HCO}_3^{-1} \rightarrow \text{UO}_2(\text{CO}_3)_2^{-2} + \text{H}_2\text{O}$ Vanadium: $\text{V}_2\text{O}_3 + \text{O} + 4\text{HCO}_3^{-1} \rightarrow 2\text{VO}(\text{CO}_3)_2^{-2} + 2\text{H}_2\text{O}$ Pyrites: $2\text{FeS}_2 + 7\text{O}_2 + 8\text{Na}_2\text{CO}_3 + 6\text{H}_2\text{O} \rightarrow 2\text{Fe}(\text{OH})_2 + 4\text{Na}_2\text{SO}_4 + 8\text{NaHCO}_3$ Calcite: $\text{CaCO}_3 + \text{CO}_2 + \text{H}_2\text{O} \rightarrow \text{Ca}_2 + 2\text{HCO}_3^{-1}$ Clay: $\text{CaClay} + 2\text{Na}^{+1} \rightarrow \text{Na}_2\text{Clay} + \text{Ca}^{+2}$ Production WellsProcessing Pregnant LixiviantStrong Base Anion IX Load: $2\text{RHCO}_3 + \text{UO}_2(\text{CO}_3)_2^{-2} \rightarrow \text{R}_2\text{UO}_2(\text{CO}_3)_2 + 2\text{HCO}_3^{-1}$

Note: Chloride or Sulfate substitute for carbonate

Elution: $\text{R}_2\text{UO}_2(\text{CO}_3)_3 + 2\text{Cl}^{-1} + \text{CO}_3^{-2} \rightarrow 2\text{RC1} + \text{UO}_2(\text{CO}_3)_3^{-4}$ Precipitation: $\text{UO}_2(\text{CO}_3)_3^{-4} + 6\text{H}^{+1} \rightarrow \text{UO}_2^{+2} + 3\text{CO}_2(\text{g}) + 3\text{H}_2\text{O}$ $\text{UO}_2^{+2} + \text{H}_2\text{O}_2 + 2\text{H}_2\text{O} \rightarrow \text{UO}_4\text{H}_2\text{O} + 2\text{H}^{+1}$ $\text{NaOH} + \text{H}^{+1} \rightarrow \text{Na}^{+1} + \text{H}_2\text{O}$ RESTORATION CHEMISTRY - Lower Eh $\text{Na}_2\text{S} + \text{H}_2\text{O} \rightarrow 2\text{Na}^{+1} + \text{HS}^{-1} + \text{OH}^{-1}$ $4\text{UO}_2(\text{CO}_2)_3^{-4} + \text{HS}^{-1} + 15\text{H}^{+1} \rightarrow 4\text{UO}_2 + \text{SO}_4^{-2} + 12\text{CO}_2 + 8\text{H}_2\text{O}$

Prepared by: Charles E. Miller [8]

Crow Butte Resources, Inc.

The plant uses 8 up flow ion exchange (IX) columns with a diameter of 14 feet (4.3 metres) and a height of 35 feet (10.7 metres). The strong based anion resin is eluted (stripped) in place eliminating the need to transfer resin. The uranium loaded resin is stripped with a sodium chloride/sodium bicarbonate eluant. Downstream of the IX columns are three precipitation tanks with agitators. The carbon dioxide in the concentrated eluant is removed by the addition of acid and precipitation of the uranium is accomplished with hydrogen peroxide (H_2O_2) followed by pH adjustment with caustic (NaOH). The precipitated uranium is then sent to a thickener. From the thickener the product is sent to a belt filter to remove excess water from the yellow cake slurry.

During January 1994, a vacuum dryer began operations which dries the yellow cake slurry to powder containing 93% U_3O_8 and less than 1 percent free moisture. The yellow cake is then shipped in 55 gallon drums to a convertor in closed trucks. Prior to 1994, the mine shipped yellow cake slurry in a special stainless steel tanker trailer. A simplified process flow sheet is included as Fig. 8. The only product recovered at the Crow Butte mine is uranium; there are no by-products. Monitoring and recording of plant processes and well field operating conditions are accomplished with a computer. The computer also controls some of the processing plant operations.

5.2.3. *Waste Water Disposal*

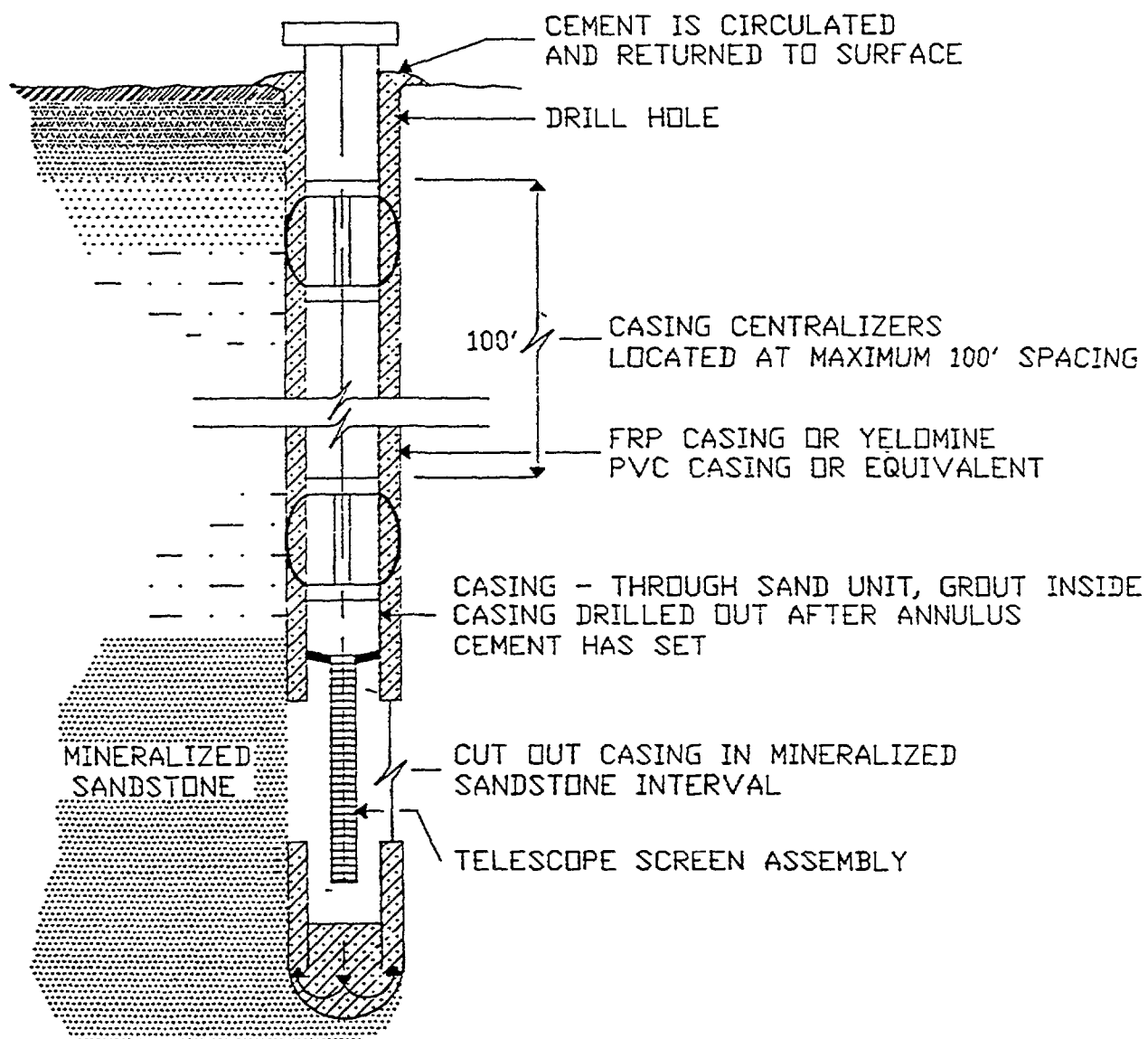
The disposal of waste water is accomplished by plastic lined evaporation ponds using a double liner system. The sources of the waste water are the well field over production, the plant bleed and the aquifer restoration liquid effluents (e.g. reverse osmosis brine). Waste water is routed from the plant to the evaporation ponds in buried PVC pipe. Two ponds were constructed prior to startup of commercial mining and an additional pond was added in 1993. Two smaller ponds are in use from the pilot plant operation. The ponds are constructed with earthen embankments and covered with a 20 mil thickness PVC bottom liner and a 60 mil thickness HDPE top liner separated by a geonet. A leak detection system located between the two liners reports any liquid coming through the top liner. During 1994, permits were obtained for both a deep disposal well and for land application of waste water. Construction for these facilities is planned for 1995 and should minimize the need for additional evaporation ponds.

5.3. **Commercial Mining Operations**

The construction of the commercial processing plant and ancillary facilities, and the installation of the first mining unit were completed in March 1991. Mining operations commenced in April 1991 with the continuous circulation of groundwater, addition of mining chemicals (carbon dioxide and bicarbonate), and the addition of the oxidant (oxygen). The first commercial well field, designated Mine Unit 1, consists of some 38 recovery and 72 injection wells. The target flow rate of 1 200 gallons per minute (76 litres per second) was achieved as the recovery wells averaged 31 gallons per minute (2 litres per second) each. A strong positive response to oxidant addition in the form of increasing uranium head grade occurred in three to four days after the oxygen system was activated.

There were only two minor problems experienced during startup that are worthy of note. One problem was the frequent plugging of the injection solution filter system (bag filters) caused by the initial surge of sediment from the 38 recovery wells turned on over a short period of time. This problem was temporary and essentially corrected itself as the production wells soon stopped producing significant amounts of sediment. The other problem was the presence of leaks in the plastic lined evaporation ponds. The ponds have double liners and a leak detection system is located between the two liners. The regulatory agencies were properly notified of the leaks and the small holes in the liner causing the leaks were quickly located and repaired. Because of the double liner system, none of the leaked waste water was released to the environment.

In March of 1992, startup operations for Mine Unit 2 commenced using a procedure of turning on only a few patterns at a time so that the formation could be pre-treated with lixiviant. By the end of July 1992 all the production and injection wells in Mine Unit 2 were in operation. Most of Mine Unit 1 remained in production as Mine Unit 2 was phased into the operation. Some of the patterns in Mine Unit 1 with very low uranium head grade were turned off. Mine Unit 2 has 52 recovery wells and 89 injection wells. At this time all the patterns in Mine Unit 2 and about 80% of the patterns in Mine Unit 1 are in production giving a total well field flow rate of about 2 400 gallons per minute (151 litres per second). The uranium head grade since startup has been lower than forecasted averaging about 66 mg/l. The problem was thought to be related to dilution of the recovery solution. To reduce the amount of dilution CBR instituted an under ream procedure for installing



CROW BUTTE RESOURCES, INC.		
CROW BUTTE PROJECT		
DAWES COUNTY, NEBRASKA		
TYPICAL MINERALIZED ZONE COMPLETION FOR INJECTION/PRODUCTION WELLS		
PREPARED BY: C.B.R.	DWN BY: TR	DATE: 11/94

FIG. 9. Well under-ream method.

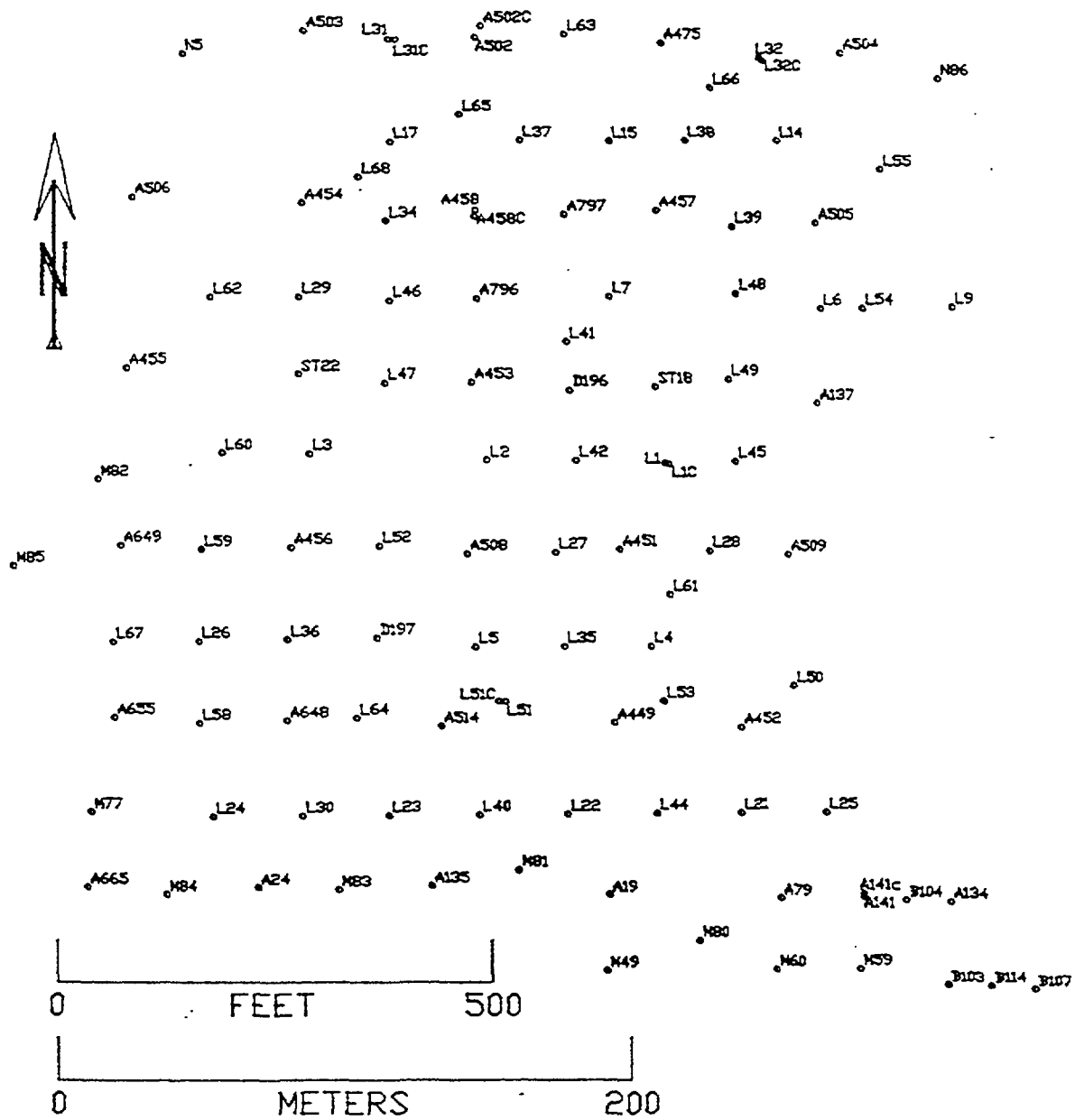


FIG. 10. Mine Unit 3, exploration and developments holes.

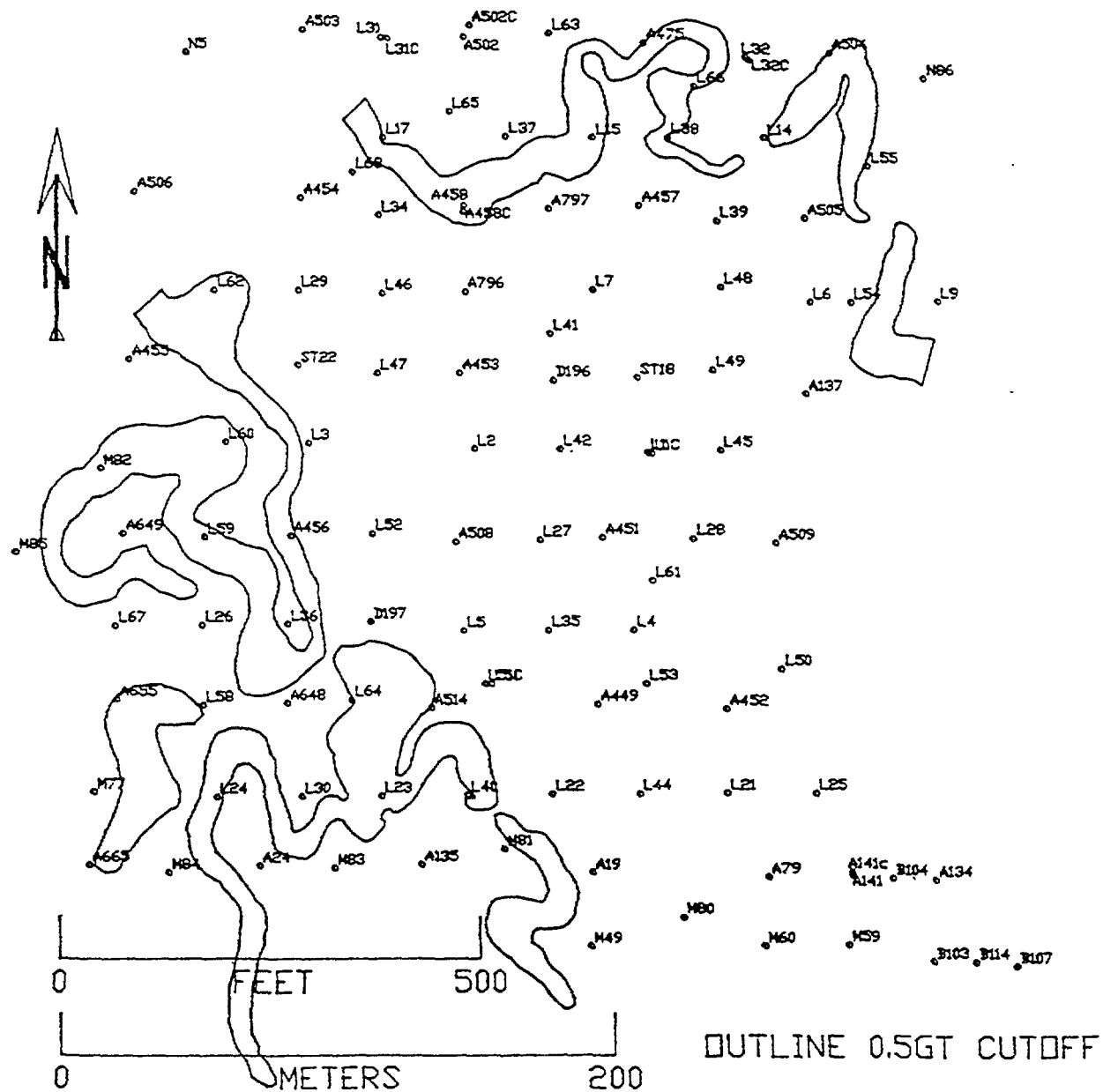


FIG. 11. Mine Unit 3, exploration and development holes (low/under yellow front).

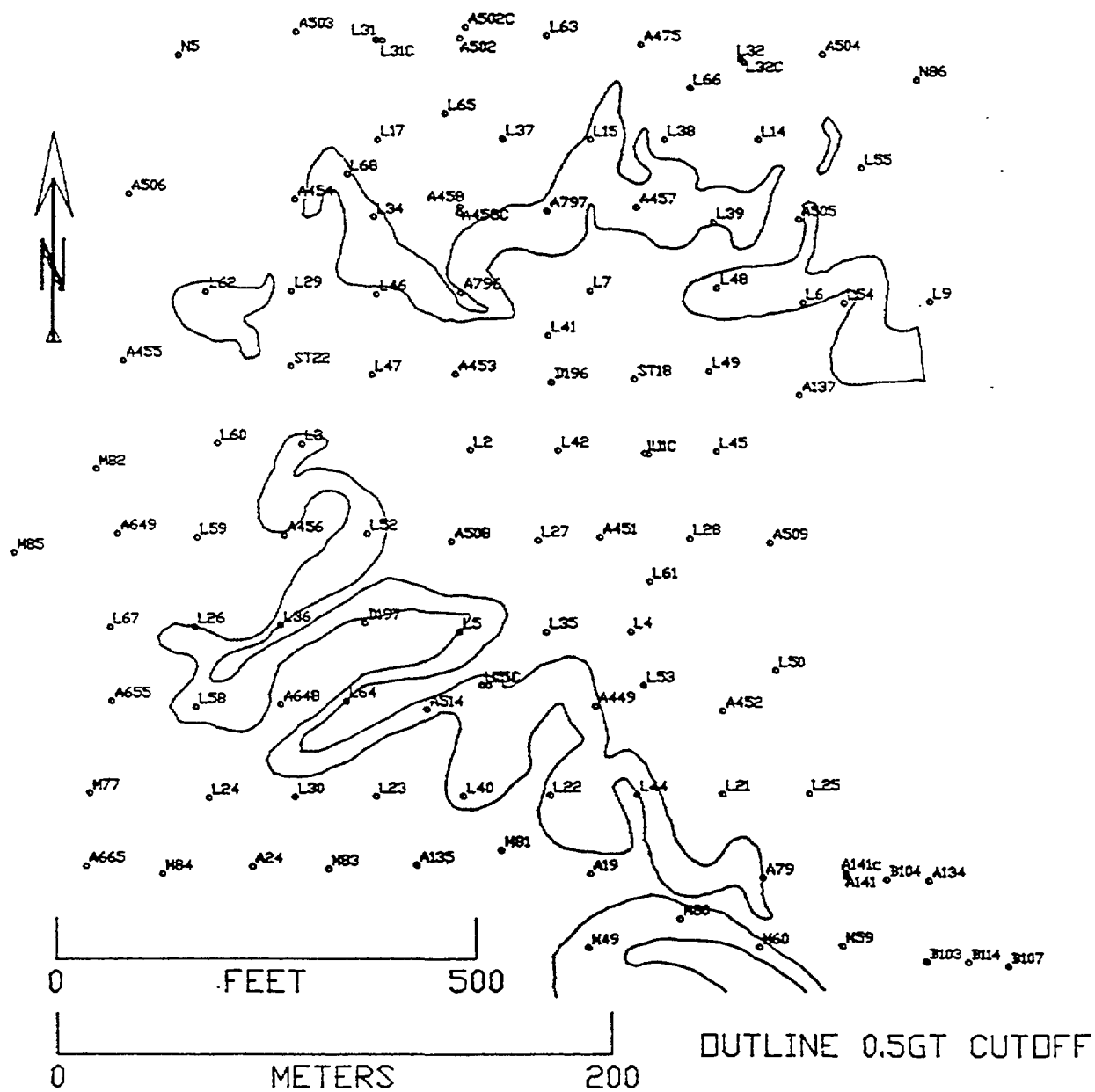


FIG. 12. Mine Unit 3, exploration and development holes (up/lower yellow front).

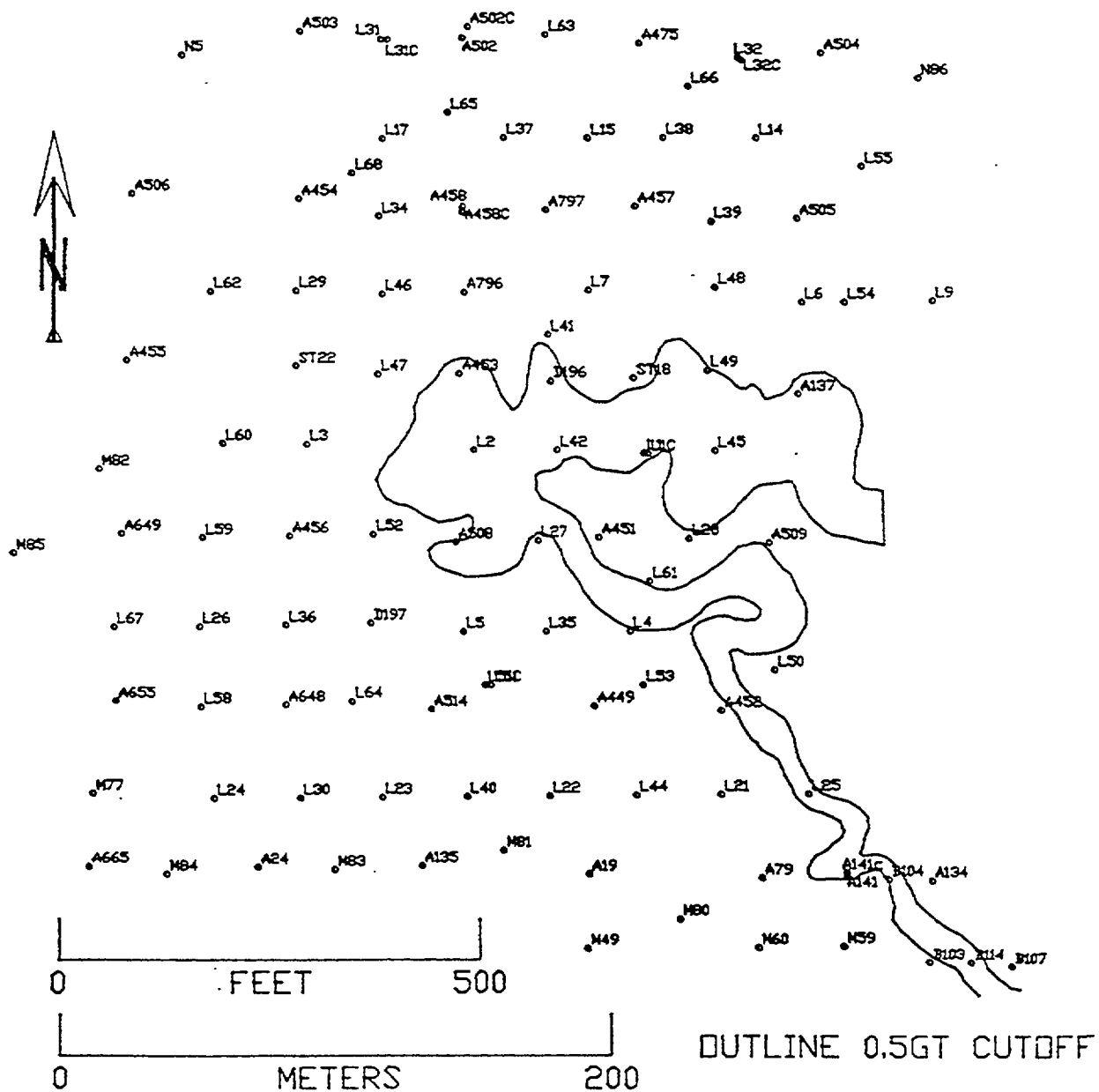


FIG. 14. Mine Unit 3, exploration and development holes (upper yellow front).

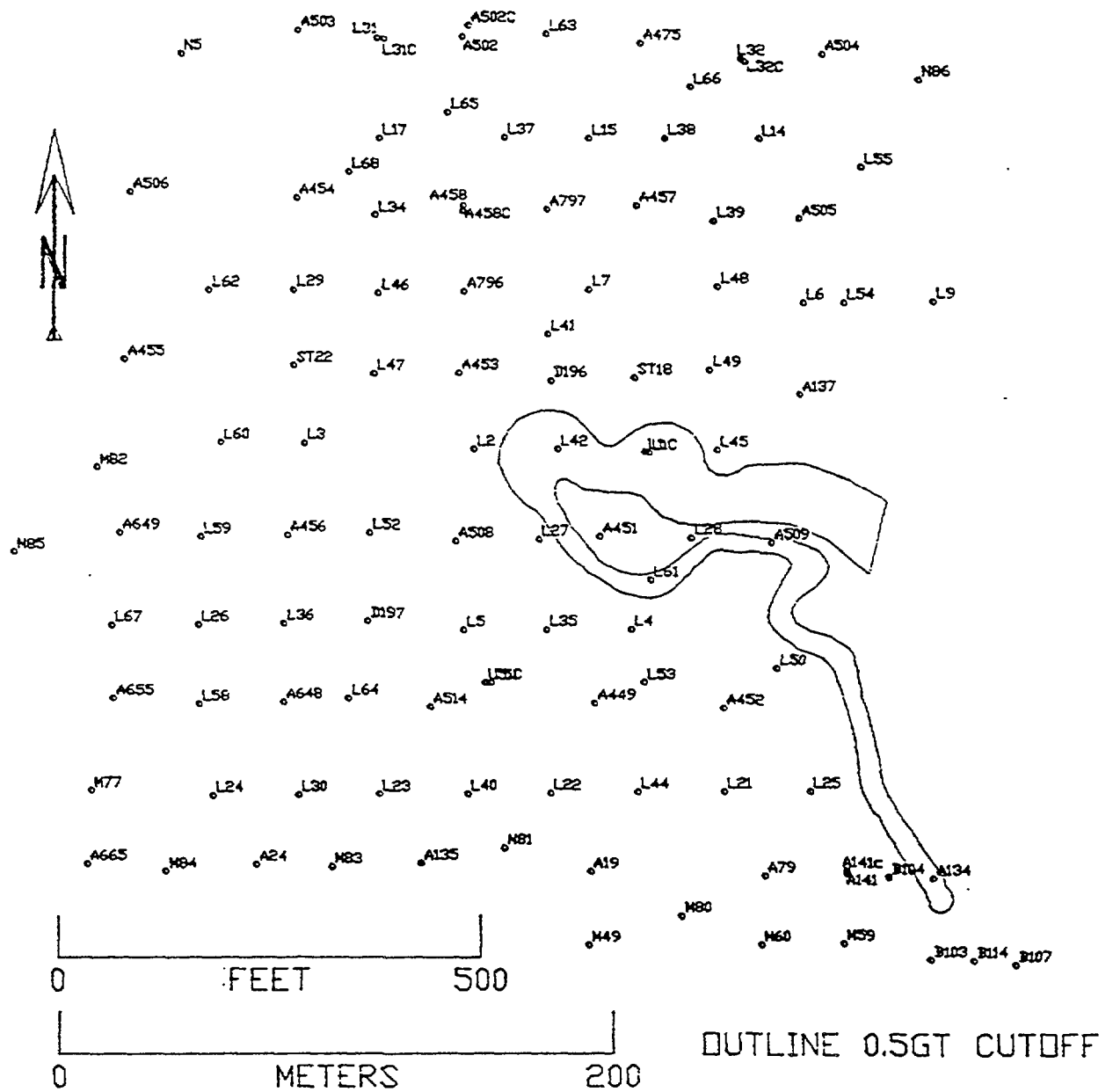


FIG. 15. Mine Unit 3, exploration and development holes (orange front).

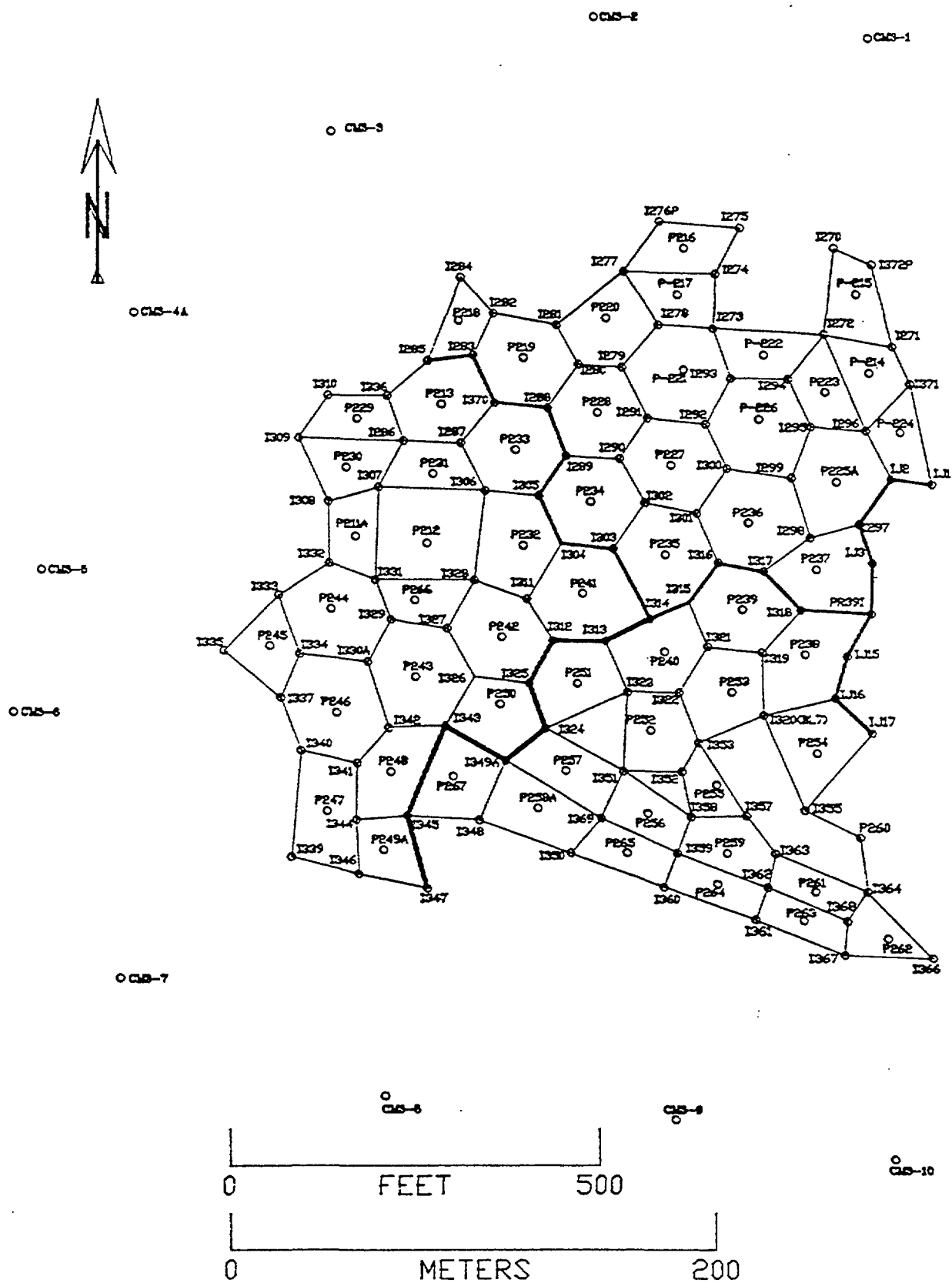


FIG. 16. Mine Unit 3.

wells during the construction of Mine Unit 3 and 4 in 1993 and 1994. it is felt that the under reaming well completion procedure will allow the project to achieve a higher average head grade thus, recovering the uranium during a shorter time period. The actual head grades are highly dependent on the grade thickness of the portion of the ore body which underlies the mine unit.

The under ream method of well completion is illustrated in Fig. 9. The drill hole is cased through the mineralization. Following determination of the screen interval the casing is cut out utilizing an under ream tool. A screen assembly is telescoped into the open interval.

Mine Unit 3 was constructed sequentially in 1992-1993. The state permit allows for the startup of a partial mine unit, if all monitor wells and baseline restoration wells for the mine unit are installed. This allows initial production from a new mine unit to come on-line sooner than it otherwise would and is a more effective utilization of resources.

Construction of Mine Unit 3 followed the drilling of 72 development holes during 1992. These development holes were in addition to 41 holes drilled during the discovery and development of the Crow Butte Mine during 1980 1991. The holes represent an approximate 100 ft. by 100 ft. (30 m by 30 m) grid over this portion of the deposit (Fig. 10). The exploration and development holes were utilized to delineate the uranium mineralization in several roll fronts within the Chadron sandstone. Five mappable roll fronts can be delineated in this portion of the Crow Butte deposit. These fronts consist of the lower and upper lower yellow, the middle and upper yellow and the orange front. The outline of ore above a 0.5 GT cutoff is illustrated on Figs 11 to 15. These fronts are subparallel, overlapping, and rise within the Chadron sandstone from west to east. The complicated nature of these fronts requires careful planning of the wellfield patterns and screened intervals of the individual wells.

Following the careful mapping of the individual roll fronts, a wellfield pattern was designed to access the ore (Fig. 16). The final design consisted of 55 patterns and 107 injection wells, a production/injection ratio of 1:1.9. Minor adjustments were made during well installation which resulted in 57 patterns with 95 new injection wells. Roll front maps were revised and ore reserves were recalculated on a front-by-front and a pattern basis using a 0.5 grade thickness cutoff, %U₃O₈-ft, (GT). All exploration and development holes and wells were used to make the final ore reserve estimate.

Mine Unit 3 was the first Crow Butte mine unit to utilize the under reaming technique of screen installation. Previously, Crow Butte utilized an integral screen completion during the installation of Mine Unit 1 and 2. The integral screen completion method resulted in larger screened intervals and thus greater dilution of solutions because of the complex nature of the Crow Butte roll fronts. The integral method required screen installation at the same time as the casing of the well. The under reaming technique allows for planning of the screened interval following the drilling of adjacent wells. This resulted in lowering the screened interval from an average of 19.6 n in Mine Unit 1 to 12.8 ft. in Mine Unit 3 or 35% less. The narrower screened interval was expected to reduce dilution and increase headgrade. An additional benefit is a smaller affected volume and thus lower restoration costs.

A comparison of Mine Unit 3 with Mine Unit 1 is shown in Table IV.

TABLE IV. COMPARISON OF MINE UNIT 3 WITH MINE UNIT 1.

Mine Unit	Patterns	Pattern Size Sq.Ft.	Screened Interval ft.	Grade Thickness %U ₃ O ₈ -ft.	Average Flow gpm	Average Head grade mg/l U ₃ O ₈ one year
1	38	10 247	19.6	1.11	30.9	68
2	57	10 284	12.8	1.26	28.8	84

The average head grade for one year improved from 68 mg/l U_3O_8 to 84 mg/l U_3O_8 , a 24% increase. The primary factor that resulted in this increase was the utilization of the under reaming technique of screen installation. Other contributing factors involved are a higher average ore grade (1.26 GT vs. 1.11 GT) and a lower average flow per well. The improvement due to the under reaming technique is estimated at 10%.

Recovery of in place reserves in Mine Units 1–3 exceeds 70% to date. Mine Unit 1 is currently in restoration and Mine Units 2, 3 and 4 are in operation. Mine Unit 5 is in construction and is planned to begin operations during mid-year 1995.

The mine operates with a compliment of 31 employees including site management. An additional 9 to 12 people operate the drilling rigs installing new well fields. The drillers are contractors and not employees of the mine operator. The plant is operated on a 24 hour basis seven days per week by two people working 8-hour shifts. The on-site laboratory is operated by two people using state of the art automated analytical equipment.

6. ENVIRONMENTAL AND RADIATION SAFETY CONSIDERATIONS

The project operates with an approximate 1/2 to 1% over production or well field bleed. This bleed accomplishes two things. First, a hydrologic cone of depression is maintained in the well field to ensure that contaminated water does not migrate out of the mining area and cause a legally defined excursion. The second reason for the bleed is to prevent the unwanted buildup of certain chemical species in the lixiviant. The bleed stream is taken out of the circuit in the processing plant and is piped to the plastic lined evaporation ponds. As of this time there have been no well field excursions at the Crow Butte mine.

The main component of the environmental monitoring program during the mining phase of the Crow Butte operation consists of the sampling of horizontal and vertical excursion monitor wells. The horizontal monitor wells are located 300 feet (91 metres) from the edge of the wellfield mining units and they are completed in the same sandstone unit containing the ore body (in this case, the basal Chadron sandstone) (Fig. 7). The vertical excursion monitor wells are located within the mining unit in the first aquifer above the ore body aquifer at a density of one monitor well per 4 acres (1.6 hectares) of well field. The baseline water quality in the monitor wells is determined prior to mining and excursion limits are established in accordance with regulatory guidelines. The excursion monitor wells are sampled and analysed once every two weeks during mining. Excursion limits are set at 20% above baseline for conductivity, sodium, sulfate, alkalinity, and chloride. If an analysis indicates that any two parameters exceed these limits or any one parameter exceeds an additional 20%, the regulatory agencies must be immediately notified and corrective actions must be initiated [9]. The leak detection system for the evaporation ponds is checked daily. The Nebraska Department of Environmental Quality monitors the operation closely for environmental compliance with a resident inspector that is on-site several times per week.

Each employee working at the mine must receive extensive training in radiation safety provided by the full time on-site radiation safety officer. The approved radiation safety programme includes personal alpha monitoring each time an employee leaves the processing plant and bioassays for uranium are performed quarterly for site personnel. Additionally, each employee must wear a personal TLD (gamma) badge which is exchanged quarterly. The radiation safety officer conducts frequent radiation checks in the plant by performing radon, radon daughters, uranium airborne particulate, and alpha/gamma surveys at numerous locations. Environmental radiation monitoring for radon and gamma levels is conducted outside of the plant building. The Nuclear Regulatory Commission (NRC) enforces the radiation safety program and conducts unannounced major inspections of the operation at least once a year.

7. AQUIFER RESTORATION AND DECOMMISSIONING

Once mining has completely stopped in a particular mining unit, the regulatory agencies require the operator to commence aquifer restoration (groundwater cleanup) operations. The goal of aquifer restoration is to return the water quality in the affected aquifer to baseline (pre mining) conditions on a parameter by parameter basis. It is recognized by the agencies that not all parameters can be returned all the way to baseline values and allowances are made for these parameters; however, the overall water quality of the aquifer after restoration is complete must be such that the water can be put to the same use as before mining. For example, if the aquifer was suitable for watering livestock before ISL mining, it must be suitable for livestock watering after restoration is complete. The operator must conduct a rigorous ground water monitoring program to document the pre mining water quality of the ore body aquifer.

At Crow Butte the operator utilizes the same aquifer restoration technique successfully tested during the pilot operation. The first phase is called groundwater sweep which involves pumping wells from the mine unit under restoration (no injection) and transferring this water to the next well field to be mined. This action draws the halo (or plume) of contaminated water just outside the well field back into the mining unit. The second phase uses water treatment to remove the contaminants from the water. The clean water (permeate) from the water treatment unit is injected back into the well field being restored. The contaminated water (brine) from the water treatment unit is sent to the evaporation ponds. At the end of the water treatment step a reductant such as hydrogen sulphide is added to the permeate being injected to stop the chemical reactions taking place in the aquifer undergoing restoration. The third phase is simply the circulation of the water through the ore body aquifer to make the quality of the groundwater homogenous.

At Crow Butte aquifer restoration of Mine Unit 1 began in 1994. It will most likely take from two to three years to complete the restoration process in Mine Unit 1. The time is necessary because active mining is occurring adjacent to Mine Unit 1. In the second phase of aquifer restoration the operator plans on using a reverse osmosis (R.O.) unit as the water treatment system. A small R.O. unit was successfully used in the pilot test programme. A 200 gpm R.O. will be installed during 1995 to assist in the restoration process.

Once all mining is completed and the affected aquifers have been properly restored, the regulatory agencies require that the site be completely decommissioned by returning the site to its pre-mining condition and use. To accomplish this all equipment and buildings must be removed from the site, contaminated residue in the ponds and any other radiative materials must be taken to a licensed disposal facility, the ponds must be backfilled and recontoured, all wells must be plugged top to bottom, and all disturbed land surface must be revegetated with native grasses. Any buried pipelines must be recovered and taken to an authorized disposal site. To ensure compliance with the decommissioning requirements and the groundwater restoration standards, the operator must maintain a reclamation performance bond of sufficient size to cover the entire cost of these decommissioning activities. For the Crow Butte mine the state of Nebraska holds the \$5.0 million reclamation performance bond submitted by the CBR. The amount of the bond is reviewed annually by both the State and NRC for its adequacy.

8. CONCLUSIONS

The Crow Butte ISL mine is the newest uranium production facility in the United States with commercial production starting in April 1991. The project went through a rigorous environmental licensing process which included a pilot testing phase to demonstrate the capability of restoring the water quality in the aquifer affected by the ISL mining. The project uses an alkaline (sodium bicarbonate) leach chemistry with oxygen as the oxidant. Hydrologically the ore is ideally located in a highly permeable sandstone formation sandwiched between two thick confining shale and clay layers. Capital development costs were kept to a minimum by acquiring very inexpensive process and

electrical equipment from closed ISL mines. Production has increased each year during the four years of commercial operations to the current production rate of 700 000 pounds per U_3O_8 per year. The increase resulted from higher wellfield flow following the addition of new mine units and by operating efficiencies. It is planned to increase production to one million pounds U_3O_8 per year as market conditions allow. The relatively low operating costs are within the range forecasted for the project. Environmentally, the Crow Butte mine is in full compliance with all State and Federal regulations. violations.

ACKNOWLEDGEMENTS

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AN INNOVATIVE JET BORING MINING METHOD AVAILABLE FOR THE HIGH GRADE URANIUM ORE UNDERGROUND DEPOSITS

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Abstract

An innovative mining method, based on the capability of a high pressure water jet to desaggregate rock, has been conceived and tested with success at the highest grade uranium ore deposit in the world, the Cigar Lake deposit in Saskatchewan, Canada. 113 tonnes of ore at 13% U were mined out by a new jet-boring mining method operated on a semi-industrial basis, in 1992 during the test mining program of Cigar Lake Project.

1. CIGAR LAKE PROJECT AND THE TECHNICAL CHALLENGE FACING THE MINING OF THE OREBODY

1.1. Cigar Lake Project

The Cigar Lake deposit occurs in northern Saskatchewan about 670 km north of Saskatoon (Figs 1 and 2).

The Cigar Lake uranium deposit was discovered in 1981 by Cogema geologists. After completion of the exploration phase the reserves of the orebody were assessed at 150 000 tonnes of uranium with an average grade about 8%.

The Cigar Lake orebody lies at a depth of 450 m (Fig. 3). The mineralization is crescent shaped lying at the unconformity between basement altered gneisses and Athabasca sandstones.

The ore thickness varies from 1 to 20 m, with a transversal extension of 20 to 100 m and a total length of about 2 km. In the vicinity of the orebody, the Athabasca sandstones are highly altered and saturated with water under hydrostatic pressure. The mineralization is enclosed in a clay envelope.

Cigar Lake Mining Corporation was established in 1985 as the operating company of the Joint Venture in which Cogema owns 36.375%. The other owners are Cameco (48.750%) Idemitsu (12.75%) and Kepco (2%).

The initial assignment to CLMC was to carry out a test mine. Its purpose was to select the appropriate mining method, for the Cigar Lake deposit.

1.2. The Test Mining Programme

The test mining program completed by the end of 1992 addressed a number of geotechnical and radiation exposure issues crucial to the successful mining of the Cigar Lake orebody:

- The reserves are located beneath 420 m of sandstone that contains groundwater directly connected to surface waters, and which thus must be controlled to prevent water inrush and contamination;
- The ore itself is surrounded by weak clay and altered rocks which display short standup time;



FIG. 1. Project locations.

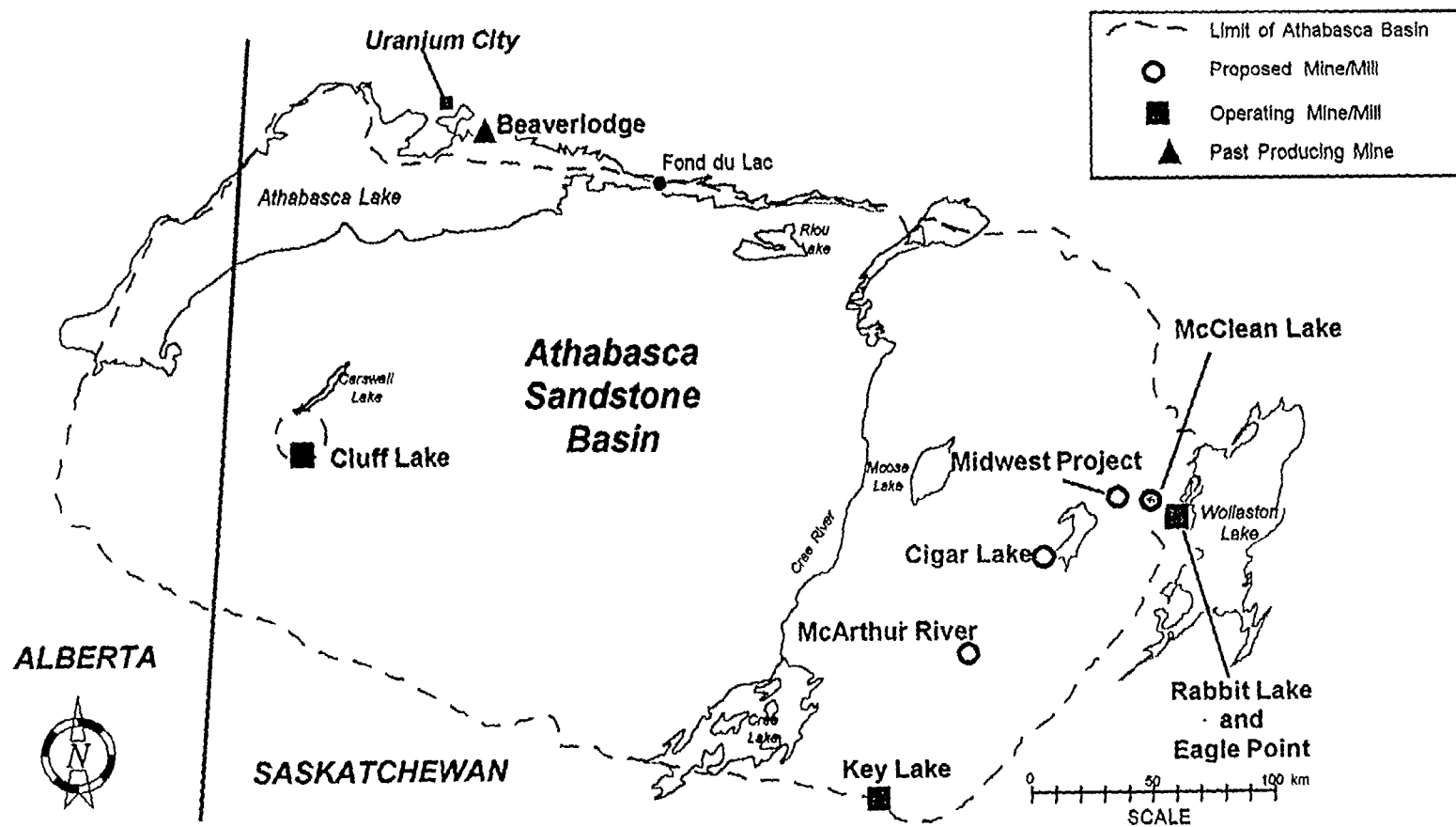


FIG. 2. Athabasca sandstone basin.

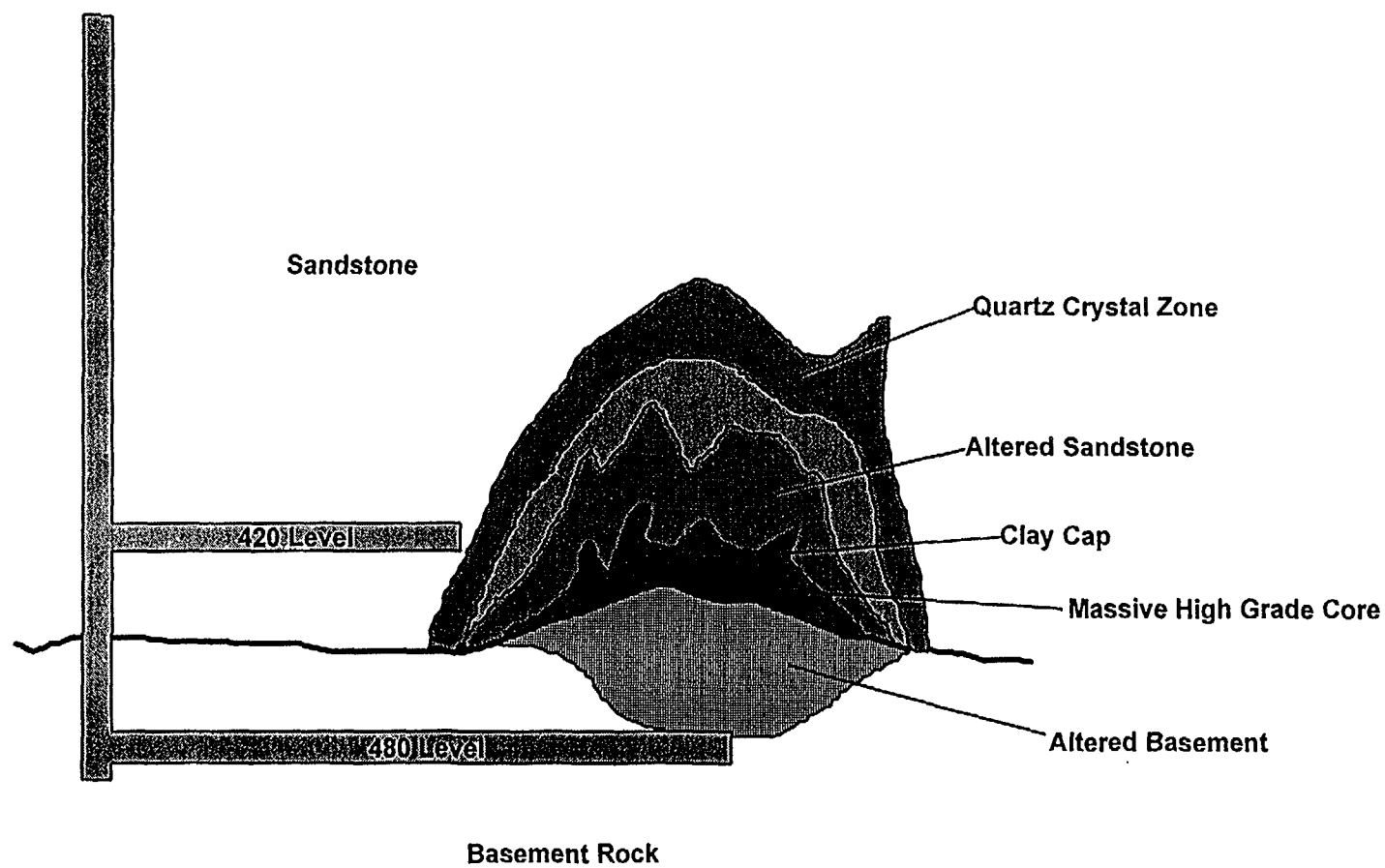


FIG. 3. Cigar Lake, descriptive cross section.

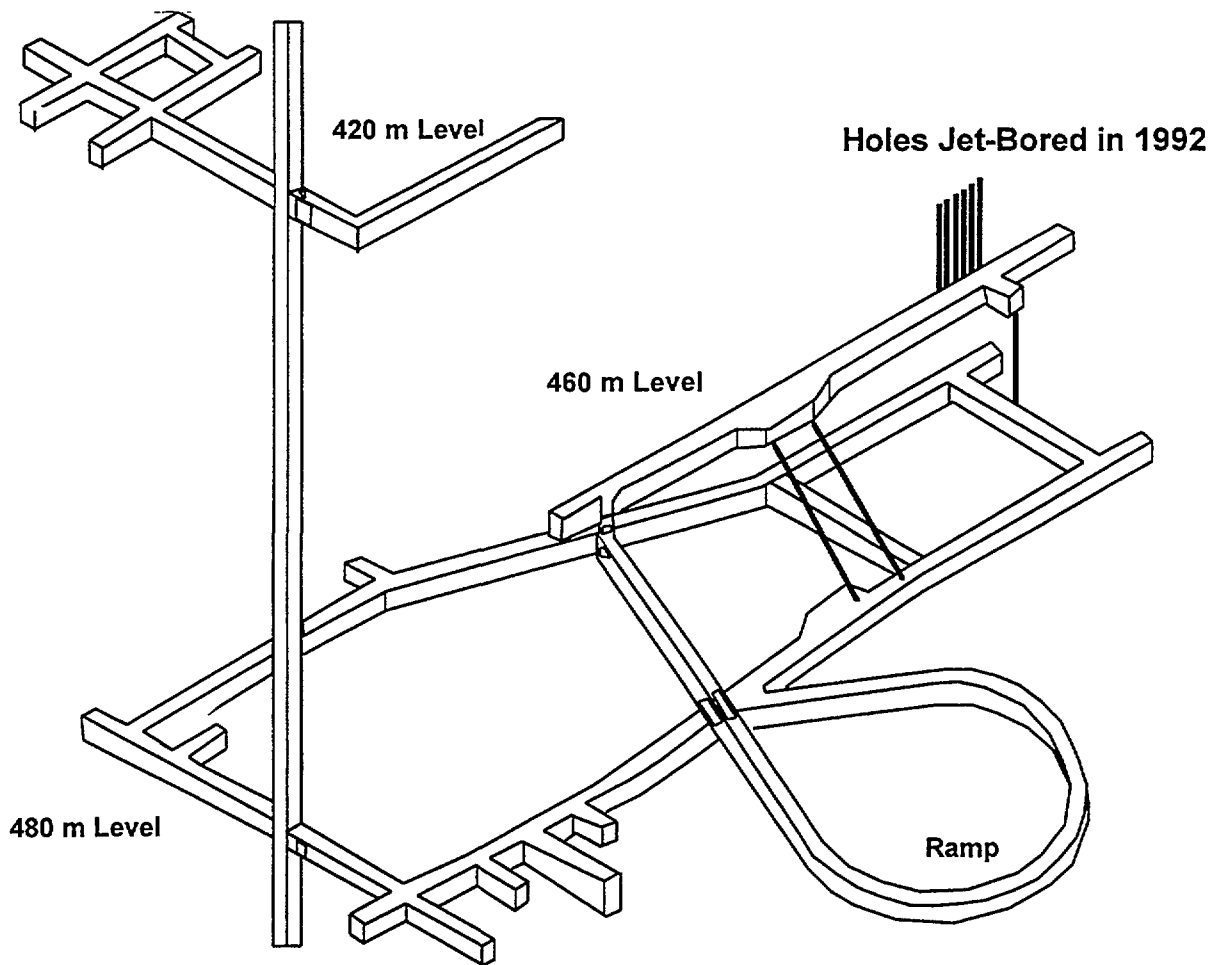


FIG. 4. Cigar Lake, test mine — general layout.

- The very high grade of the ore requires that non-entry remote control mining methods be used to provide radiation protection.

1.3. Results of the Test of the Jet Mining Method at Cigar Lake

Three field tests were completed in 1992, on a semi-industrial basis, mining out 113 tonnes of ore at a grade of 13.2 % U, from 6 different holes (Fig. 4).

The main goals were:

- to optimize the production rate and the hole size,
- to assess the backfill conditions.

The main following jetting parameters were available for variations:

- water temperature from +15°C to +30°C,
- jet pressure, from 50 MPa to 90 MPa,
- rotation, translation speeds from 5 to 20 RPM, and 5 to 15 cm/mn.

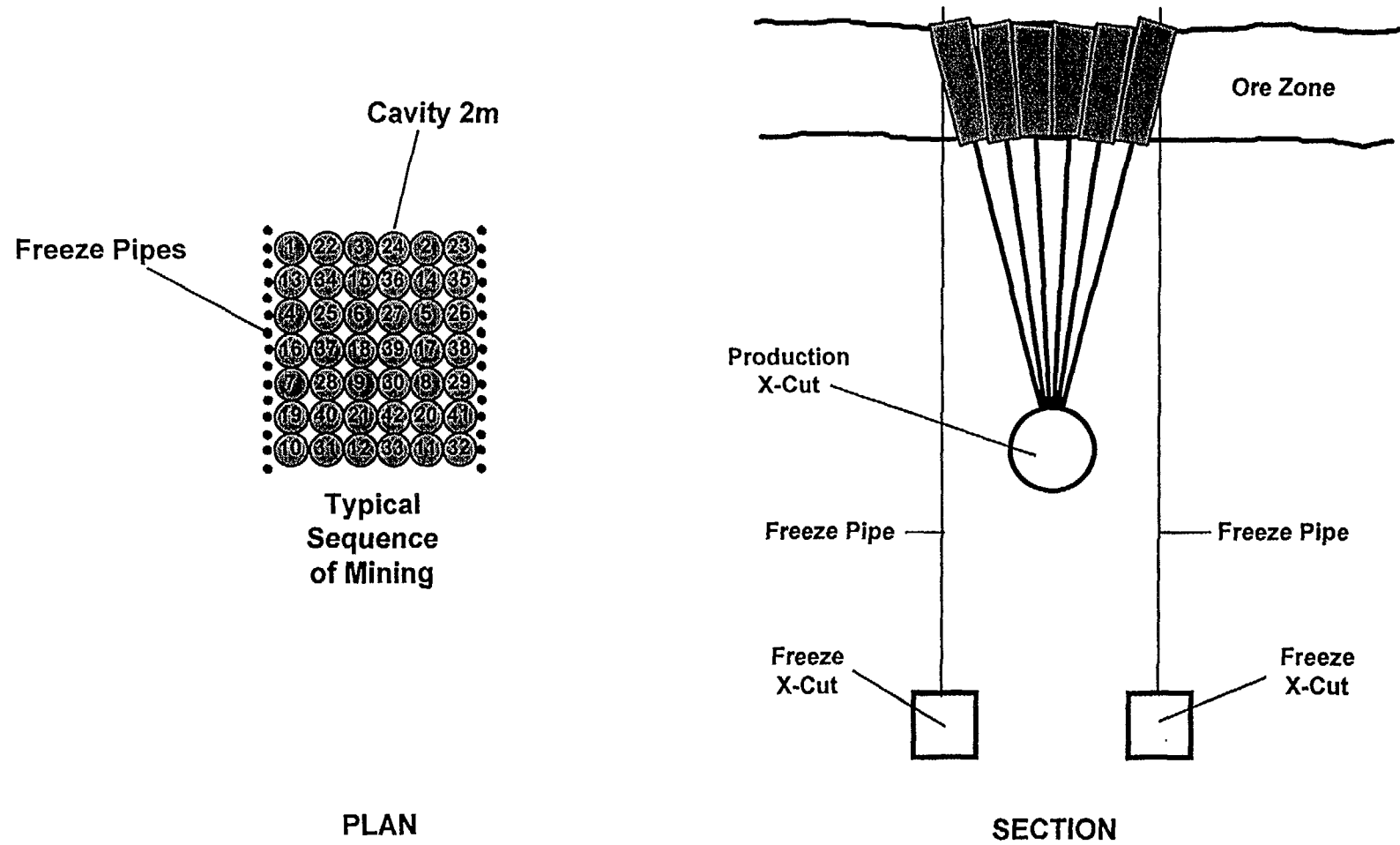


FIG. 5. Jet-boring mining method.

The results were:

- Ore was produced from each hole and massive pitchblende was successfully extracted during the test, Production rate varied from 4 to significantly more than 10 tonnes per hour, A cavity diameter of 2.5 metres was attainable.
- Stability of the frozen massive orebody was not impacted by the water jet beyond the cavities exploited. Both gamma and radon daughters exposure were low during the test.

2. THE JET-BORING MINING METHOD

2.1. Panel Layout and Mine Development

2.1.1. Panel Layout

The jet-boring mining method in frozen ground (Fig. 5) requires the development of two different levels beneath the orebody:

- The lower level is the freezing level from which all freezing operations are conducted;
- The upper level is the production level dedicated initially to the drilling of the casing holes and later to the mining of the ore.

The mining is conducted in approximately 12 metre wide mining panels in the length of the deposit. The width of the panels depends on the thickness of the orebody and provides for an adequate pillar between crosscuts while maintaining a reasonable ground freezing time.

Depending on the ground conditions and the potential for a significant water inrush at the unconformity, a 10 to 15 metre crown pillar between the production level and the unconformity is required. The freeze level is 10 to 15 metres below the production level. For each panel there are 2 parallel freeze crosscuts, 2 metres apart and 1 production crosscut. The production crosscuts are offset halfway between the freeze crosscuts

The freeze level and production level crosscuts are connected to perpendicular drifts.

2.1.2. Development of the Freezing Level

The development (Fig. 6.1) is first carried out on the freezing level, using either conventional drill and blast mining method or a shielded roadheader capable of erecting concrete liner segments when rock conditions appear to be very poor.

In the case of the Cigar Lake deposit, the freezing level occurs entirely in the basement rock. Although the ground is not frozen, water inflow remains low due to the low hydraulic conductivity of the basement rock.

Vertical freeze pipes (Figure 6.2) are installed vertically every 2 metres along the boundaries of a mining panel in a pattern which ensures complete freezing of the orebody. Freezing is also extended above the orebody to ensure water control at the top. The drill equipment incorporates a preventor system to control possible water inrush, as the ground is not frozen during this phase. Freezing is achieved by circulating brine at a temperature of about -30°C through the freeze pipes. The frozen ore and surrounding rock is maintained at a temperature around -15°C.

The freezing time to freeze a panel may range from 9 months to 1 year.

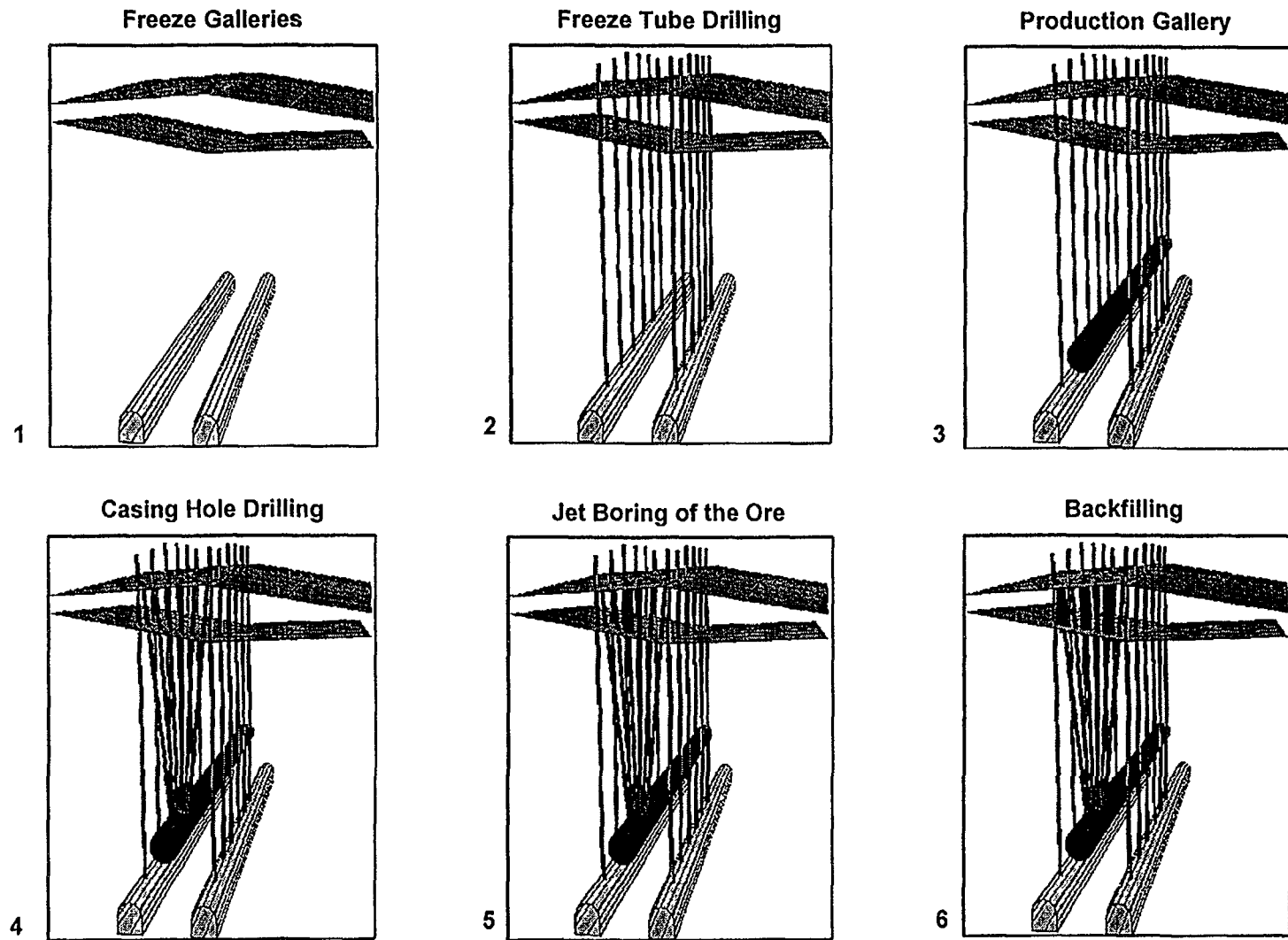


FIG. 6. Typical mining sequence.

2.1.3. Development of the Production Level

After freezing of the panel, the production drift (Fig. 6.3) is driven along the length of the panel. This is done in frozen ground and therefore the drift has to be circular to ensure its stability: a roadheader is used in the frozen ground, and a concrete liner is installed to support the ground. The finished production drift will have a diameter of approximately 4.5 metres.

The layout of the development level has to take into account the geometry of the orebody, the geological environment, and the expected ground conditions.

After completion of the production crosscut, a drilling machine drills long and large diameter holes, and installs casings.

Casing holes are drilled (Fig. 6.4) from below the orebody to provide the jet-boring mining system with access to the ore zone.

Six casing holes are drilled in a fan pattern, every 2 metres. The holes are equipped with steel casings up to the bottom of the orebody. These casings provide the path along which the mined material travels to the bottom of the hole for collection. All casing holes in a panel are completed prior to the commencement of the jet-boring operations.

2.2. Mining and Backfilling a Cavity

2.2.1. Mining of an Elementary Cavity

The jet-boring machine is located beneath the orebody in the production gallery (Fig. 7). The jet-boring equipment includes a high pressure water pump, to supply water to the jet-boring head; a suitable slurry handling system, to transport the cuttings to the grinding area; a backfilling system, to backfill the mined-out cavities while jet-boring proceeds onto the next hole.

The system incorporates a preventor system to control possible water inrush, and a decontamination capability including a rod cleaning system. An in-hole cavity survey tool is also included. The system provides adequate radiation protection.

When all the casing holes are drilled, the jet-boring machine begins to operate.

The jet-boring rods are introduced in the cased holes and extended into the orebody. The jet-boring system mines the ore by cutting it with a high-pressure rotating water jet located at the top of the rods. The jet of water breaks the material away from the wall of the cavity and the material falls to the base of the cavity. The ore slurry flows through the annulus between the casing and the rods, and then directed through the water flush cone crusher located in the production drift.

The water jet is capable of cutting a cavity of a few metres in the ore, but productivity decreases when the diameter increases. The cavity is cylinder-like (Fig. 6.5).

2.2.2. Ore Handling — Crushing — Hoisting

The jet-boring mining method produces a dilute slurry. It is therefore advantageous from a safety and operational point of view, to handle and hoist the ore stream as a slurry. The crushing and grinding circuits are located underground in order to fine grind the ore and densify the slurry into a suitable form prior to hoisting.

The ore is pumped as a slurry from underground to storage tanks located on surface in an ore loadout building. Vertical cased holes are used to hoist the ore slurry to surface.

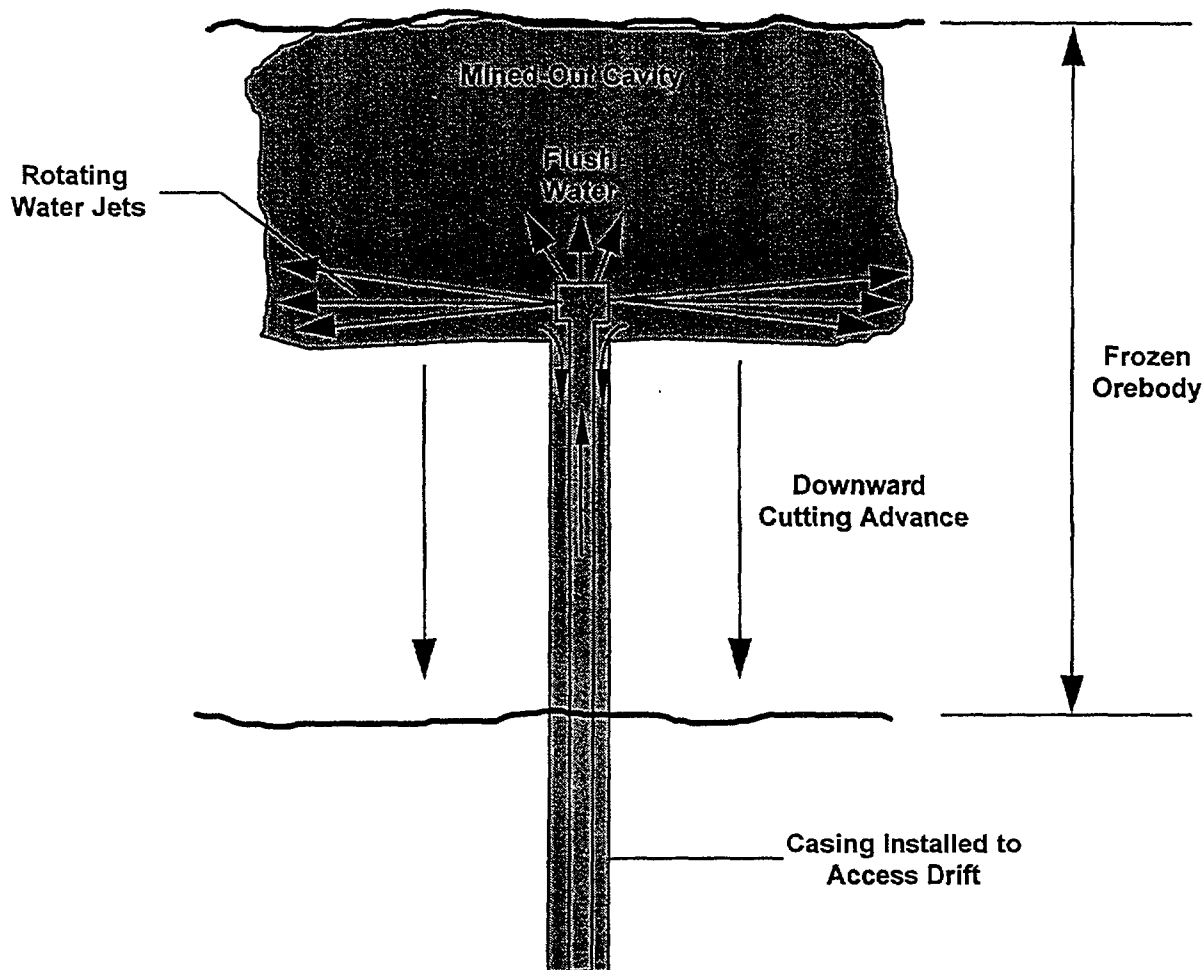


FIG. 7. Jet-boring mining method, basic concept.

2.2.3. Backfilling

Immediately after removal of the ore, the jet-boring excavation is backfilled (Fig. 6.6) with a concrete mix to provide the support required for the mining of adjacent cavities, and to ensure global stability.

The concrete mix uses a high quality cement and is specifically designed for backfilling in a frozen cavity. It is intended that the concrete strength is sufficient to resist cutting by the jet-boring machine during the mining of adjacent cavities.

Backfilling is achieved by pumping concrete through the casings. The concrete pump is part of the jet-boring machine. A small diameter air bleed pipe is first installed from the bottom of the casing hole to the top of the cavity. The concrete is then introduced through a slick line connected to the bottom of the casing. The concrete fills the cavity from the bottom up to the top while pushing the trapped air through the bleed pipe.

Mining of an area adjacent to an already backfilled cavity is possible when the concrete backfill has set:

Overall the recovery of the ore is nearly 100%, not considering the apparent losses of ore in the design (Fig. 8): in reality the water jet cuts everything between the backfilled cavities.

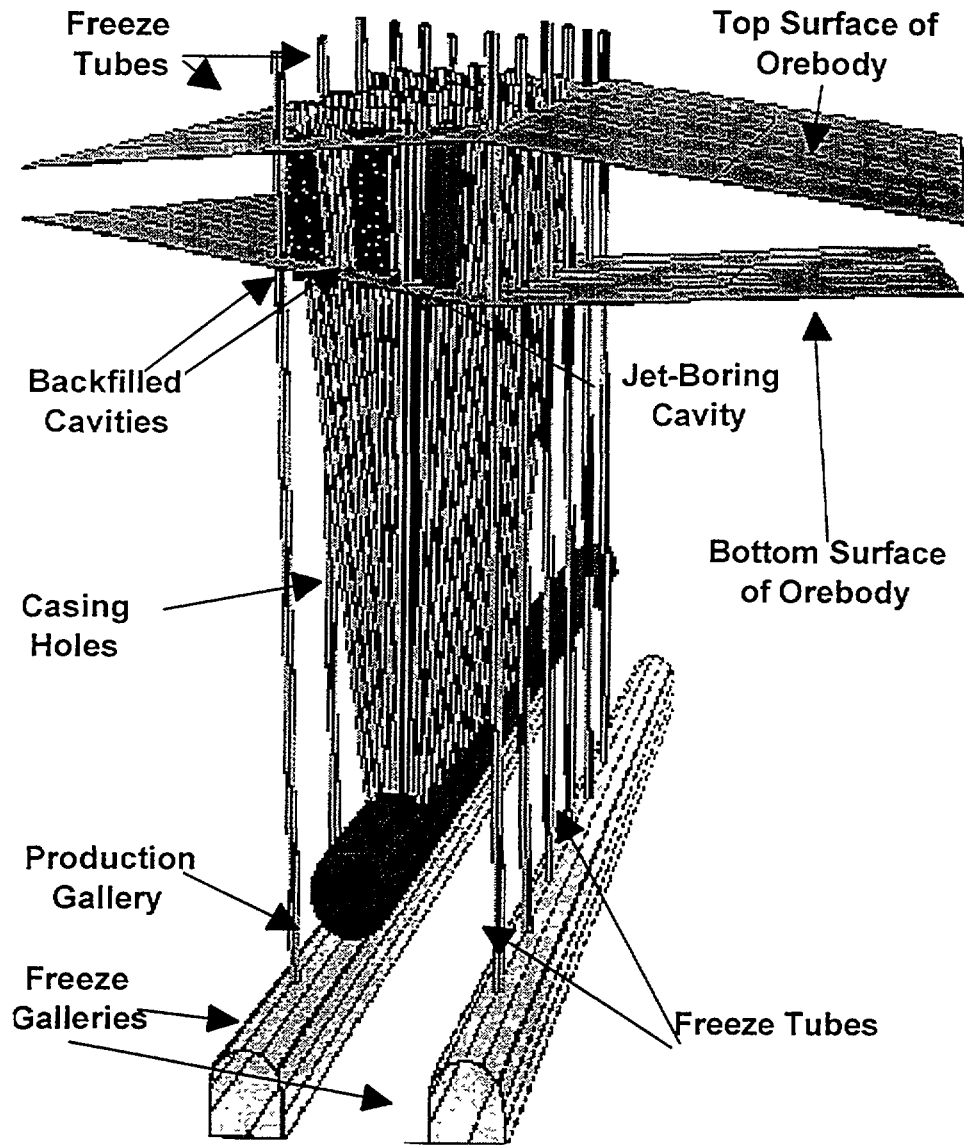


FIG. 8. Jet-boring mine layout.

As mining panels are mined out, the corresponding production and freezing drifts are backfilled to ensure ground stability and to help manage the ventilation circuit (Fig. 9).

3. ORIGINALITY AND ADVANTAGES OF THE METHOD

What makes the method original is the combination of: Freezing of the deposit, by panel, from underneath prior to mining, Mining with a high pressure water jet from a production level below the ore deposit, Handling of the ore in a slurry form contained in pipes using gravity to extract the ore from the cavity, and then hydraulic hoisting up to surface.

The advantages of the method are as follows:

- Ground freezing ensures stability of the weaker rock and controls water inflows and radon emanation during development and mining.

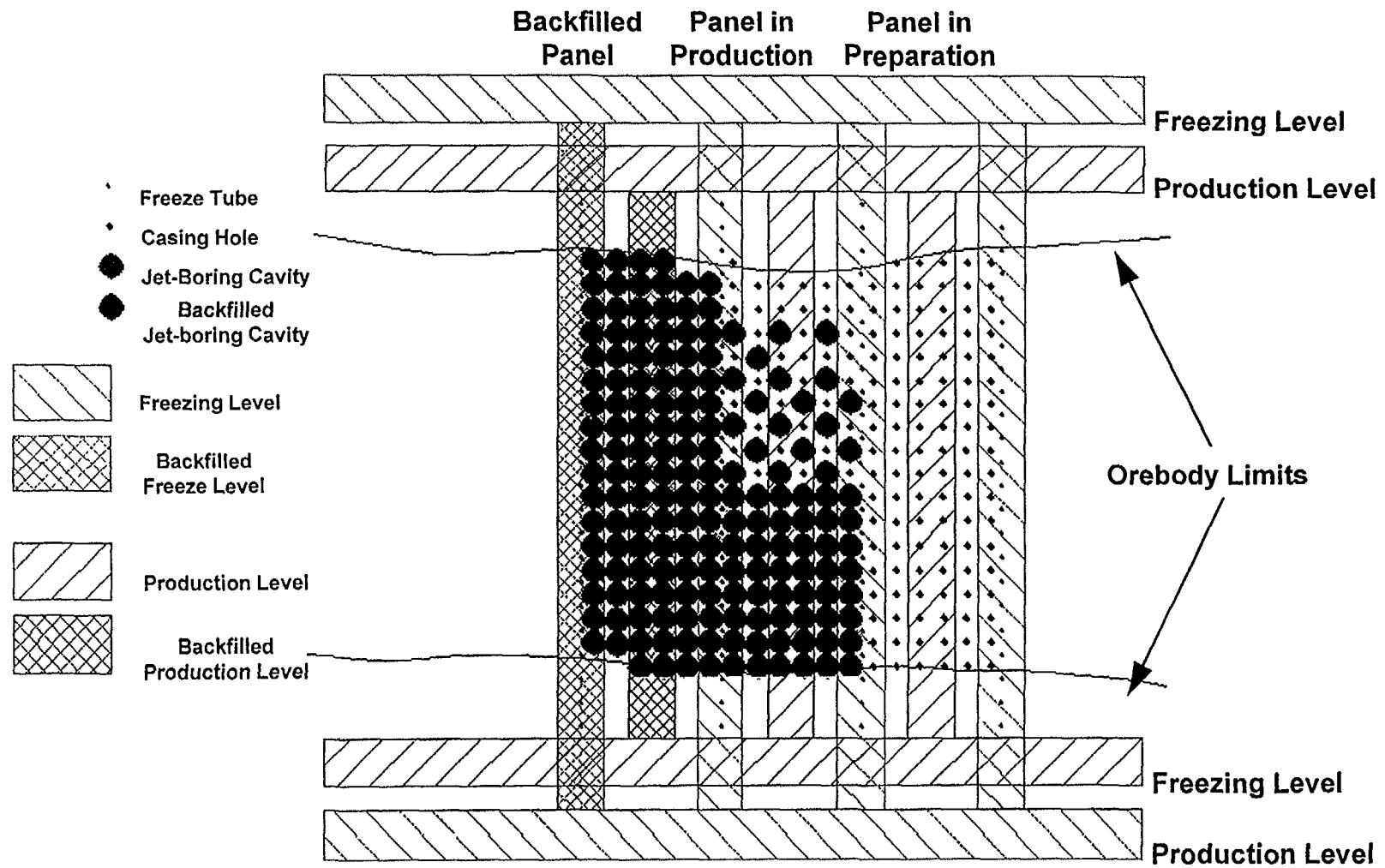


FIG. 9. Jet-boring successive mining panels.

- The method is a non-entry mining method, allowing for the mining of cavities away from drifts located below the orebody. Workers are separated from the ore by a thick layer of barren rock which shields them effectively from gamma exposure.
- Ore is collected in a slurry form at the bottom of the cavities and is contained within cuttings collection system and pipes ensuring isolation from the working environment. Jet-boring is a wet process therefore dust generation is eliminated.
- The water jet is able to extract the ore next to a backfilled cavity without significant dilution of the ore with concrete.
- Due to the ability to mine in a fan pattern with the jet-boring mining method there is a minimal amount of drifting which is required.

4. CRITERIA FOR THE APPLICATION OF THE METHOD AND COMPONENTS OF THE MINING OPERATING COSTS

The criteria to consider, in order to determine if the method is applicable to a given ore deposit are essentially:

- the ability to cut the frozen ore with a high pressure water jet,
- the ability to develop the freezing level below the ore deposit,
- the geometrical features of the deposit and the determination of the appropriate size of the crown pillar between the orebody and the freezing and production levels,
- the mining operating costs.

The mining operating costs depend in particular, of the geometrical features of the ore deposit: length of the panels, thickness of the mineralization, etc....

Assuming for example,

- Mineralization thickness: 10 metres
- Panels length: 100 metres
- Panels width: 12 metres,
- Distance between the production level and the orebody: 10 metres,
- Distance between the freezing level and the orebody: 20 metres,

The order of magnitude of the direct mining operating costs are on a percentage basis:

- Galleries: 15%
- Freezing: 18%
- Casing holes: 40%
- Jet-boring/backfilling: 27%

Drilling and equipment of the freezing and casing holes is the major component of the direct mining cost, and depends on the efficiency of the drill.

5. CONCLUSION

As described above, the jet-boring mining method is an appropriate method applicable to the Cigar Lake deposit since it best addresses all the technical challenges characterizing the deposit. It obviously can be applied to the mining other orebodies and to the creation of cavities for various purposes.

This method has also been chosen to mine the Midwest ore deposit, which is currently being studied by the Midwest Joint Venture, with Cogema as the operator.

This innovative mining method was patented in 1994 jointly between Cogema and Cigar Lake Mining Corporation.

APPLICATION OF RADIOMETRIC ORE SORTING TO KALIMANTAN ORES (KALAN PROJECT)

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Abstract

For application of radiometric sorting, three conditions should be satisfied: a sufficient heterogeneity of uranium minerals among ore particles, a hard rock producing few fines through crushing, a constant radiometric equilibrium between uranium and radium in the ore. Information from BATAN people indicate that the two first conditions are fulfilled. The last one was confirmed at COGEMA - SEPA in France. A sample of seven tons was sent to France. An important series of tests was carried out. The results indicate that 75% is amenable by radiometric sorting. The enrichment ratio is near 4 with an uranium loss of nearly 10%. This means a recovery of 65% of the total uranium. The fraction which cannot be treated by radiometric sorting (the fine particles < 30 mm) can be upgraded by gravimetric methods (KNELSON concentrator): 18% of the 25% are recovered. It is the best results we have ever obtained from radiometric sorting on uranium ores.

1. INTRODUCTION

1.1. KALAN Project

Indonesian authorities intended to recover uranium and rare earths in KALIMANTAN. It is called the KALAN Project. These complicated operations decided BATAN people who are in charge of the project to ask the help of IAEA for the feasibility study.

Several missions of geologist, mining and ore processing experts took place, especially in 1990 and 1992.

Many years ago, French exploration in the KALAN area had shown a very intricate geology. The BATAN people have worked again in this area. This work allowed a better knowledge of the geology but all is not clear, especially the boulders formation containing the rare earths.

For uranium, the minable part is located at EKO REMAJA inside the KALAN area. It is with the ore coming from EKO that the radiometric sorting tests were carried out at Bessines.

Rare earths boulders were also found in a deposit near RIRANG, inside the KALAN area. They contain monazite i.e. rare earths phosphates associated with uranium.

1.2. Ore Processing activities

The ore processing activities allowed to measure the complexity of the problem. Many people came during the last ten years to France for training, especially at the Bessines mill and laboratory. Samples of ores were available in small quantities to carry out laboratory tests only.

Many of these samples were used for leaching, solid/liquid separation, purification and yellow-cake precipitation tests. All the information collected on ore processing did not allow to prepare a coherent flowsheet. There was a too large variability in the ores.

Since 1989, we have focused our work on EKO and RIRANG deposits. It seems possible to product a yellow-cake in the KALAN area using classic ore processing techniques. A large pilot is

available in the area at Lemajung. This pilot must be improved for working in good conditions. This was one of my recommendation in 1992. A little uranium production could be achieved without large investments.

RIRANG boulders could be also processed in this installation to recover uranium. The rare earths are not leached. They stay in the solid residue. In spite of the absence of thorium, the radioactivity of leach residue is high.

To reduce the operating costs, we have to upgrade the ore. The ore processing costs are more or less proportional to the ore tonnage rather than to the quantity of uranium.

The purpose of this paper is to present the results obtained by radiometric ore sorting on EKO REMAJA ore and by gravimetry for the ore non amenable by ROS.

2. EVALUATION OF SORTABILITY AND TESTS METHODOLOGY

For the application of radiometric sorting, three conditions should be satisfied:

- a sufficient contrast weight of uranium minerals among ore particles,
- a hard rock producing few fines through crushing,
- a constant radiometric equilibrium between uranium and radium in the ore.

From field operations, it appears that the first two conditions seems to be fulfilled. The uranium-radium radioactive equilibrium measurements were carried out at Bessines on each size fraction (+80 mm, 50-80 mm, 30-50 mm).

On each size fraction, around thirty samples were counted, weighted and then ground for chemical analysis. A correlation line was then drawn between the ratio shocks number/weight and the uranium chemistry results.

The correlation line is slightly different for each size fraction. However the correlation is quite satisfactory as it appears in the following table:

Size fractions	Number of chips	Correlation coefficient
+ 80 mm	28	0.975
50-80 mm	26	0.95
30-50 mm	26	0.98

The three preliminary conditions being fulfilled, the tests were operated as follows.

- The ore was screened on large screens with shovels to obtain four fractions:
 - fraction 1: +80 mm,
 - fraction 2: 50-80 mm,
 - fraction 3: 30-50 mm,
 - fraction 4: -30 mm.

The three coarser fractions were washed and the fines were then mixed with the minus 30 mm fraction. After division, a representative sample of this last fraction was then screened on laboratory screens.

- On each coarse fraction, static gamma radiation and mass measurements were carried out using:
 - 779 chips from the 30-50 mm fraction,
 - 549 chips from the 50-80 mm fraction,
 - 337 chips from the +80 mm fraction.
- The weight of all the chips on each fraction was respectively:
 - fraction : 30-50 mm 82.9 kg,
 - fraction : 50-80 mm 219.8 kg,
 - fraction : +80 mm 4.03.8 kg.

Each of these chips were put in a lead vault. Gamma emission was measured with a Geiger type counter disposed at the bottom of the vault through a hole in the lead vault.

Each day, for calibration, a measurement was made without any chip in the vault and the result was subtracted from the results obtained with the chips. An accurate value of the gamma radiation for each chip is now well known. Each of these chips were then weighted on a laboratory scale.

3. CHARACTERIZATION OF EKO REMAJA ORE COMING FROM KALIMANTAN TO CARRY OUT RADIOMETRIC SORTING IN FRANCE

The results of the uranium content, uranium distribution and weight per size fraction is given in Table I.

TABLE I. URANIUM/SIZE ANALYSIS

Ore size	Weight	Uranium analysis ppm	Uranium distribution %
> 80	14.0	3 104	21.2
50 - 80	27.0	2 225	29.3
30 - 50	23.6	2 161	24.9
25 - 30	8.0	1 592	6.2
15 - 25	11.4	1 224	6.8
8 - 15	9.7	1 448	6.8
5 - 8	1.9	1 244	1.2
2 - 5	2.6	1 224	1.6
1 - 2	0.4	1 776	0.3
0.5 - 1	0.3	2 352	0.3
0.25 - 0.5	0.2	3 240	0.3
0.1 - 0.25	0.2	4 092	0.4
< 0.1	0.7	2 124	0.7
TOTAL	100.0	2 052	100.0

TABLE II. CHARACTERISTICS OF THE CLASSES

	30 - 50 mm			50 - 80 mm			+ 80 mm		
Uranium class ppm	Weight %	Uranium content ppm	Uranium distribution %	Weight %	Uranium content ppm	Uranium distribution %	Weight %	Uranium content ppm	Uranium distribution %
< 50	44.3	19	0.4	34.6	19	0.3	17.1	21	0.1
50 - 500	19.1	201	1.8	17.9	212	1.7	14.2	185	0.9
500 - 1 000	5.9	779	2.1	6.9	720	2.2	14.4	704	3.2
1 000 - 2 000	10.5	1 438	7.0	16.7	1 464	11.0	17.0	1 414	7.8
2 000 - 3 000	4.1	2 469	4.7	6.9	2 297	6.9	7.9	2 439	6.2
3 000 - 4 000	2.9	3 451	4.6	3.6	3 481	5.7	6.3	3 388	6.8
4 000 - 5 000	3.1	4 415	6.3	2.6	4 467	5.2	5.1	4 434	7.3
5 000 - 10 000	4.5	7 290	15.2	5.0	6 703	15.0	10.6	6 707	22.9
> 10 000	5.6	22 342	57.9	6.0	19 270	52.0	7.4	18 771	44.8
TOTAL	100.0	2 161	100.0	100.0	2 225	100.0	100.0	3 104	100.0

TABLE III. URANIUM/SIZE ANALYSIS (30–50 mm size fraction (U = 2161 ppm)

Cut off grade U ppm	REJECT			ACCEPT		
	Weight %	Uranium content %	uranium distribution %	Weight %	Uranium content (ppm)	Uranium distribution %
50	44.3	19	0.4	55.7	3 865	99.6
500	63.4	74	2.2	36.6	5 777	97.8
1 000	69.3	134	4.3	30.7	6 737	95.7
2 000	79.8	305	11.3	20.2	9 462	88.7
3 000	83.9	411	46.0	16.1	11 280	84.0
4 000	86.8	513	20.6	13.2	13 001	79.4
5 000	89.9	647	26.9	10.1	15 636	73.1
10 000	94.4	964	42.1	5.6	22 342	57.9

TABLE IV. ORE SORTING TEST RESULTS (50–80 mm size fraction (U = 2161 ppm))

Cut off grade U ppm	REJECT			ACCEPT		
	Weight %	Uranium content %	uranium distribution %	Weight %	Uranium content (ppm)	Uranium distribution %
50	17.1	21	0.1	82.9	3 740	99.9
500	31.3	95	1.0	68.7	4 475	99.0
1 000	45.7	287	4.2	54.3	5 747	95.8
2 000	62.7	593	12.0	37.3	7 325	88.0
3 000	70.6	799	18.2	29.4	8 638	81.8
4 000	76.9	1 011	25.0	23.1	10 070	75.0
5 000	82.0	1 224	32.3	18.0	11 667	67.7
10 000	92.6	1 852	55.2	7.4	18 771	44.8

TABLE V. ORE SORTING TEST RESULTS (+80 mm size fraction (U = 3104 ppm))

Cut off grade U ppm	REJECT			ACCEPT		
	Weight %	Uranium content %	uranium distribution %	Weight %	Uranium content (ppm)	Uranium distribution %
50	17.1	21	0.1	82.9	3 740	99.9
500	31.3	95	1.0	68.7	4 475	99.0
1 000	45.7	287	4.2	54.3	5 474	95.8
2 000	62.7	593	12.0	37.3	7 325	88.0
3 000	70.6	799	18.2	29.4	8 638	81.8
4 000	76.9	1 011	25.0	23.1	10 070	75.0
5 000	82.0	1 224	32.3	18.0	11 667	67.7
10 000	92.2	1 852	55.2	7.4	18 771	44.8

TABLE VI. ORE SORTING TEST RESULTS (Cumulated results on 30-50 mm, 50-80 mm, +80 mm size fractions (U = 2392 ppm))

(These cumulated results suppose the three fractions are treated simultaneously on three different channels with their respective characteristics, specific weights and uranium contents as resulting from table I, for the same cut off grade, and with their specific ore sorting characteristics as resulting from tables III, IV and V).

Cut off grade U ppm	REJECT			ACCEPT		
	Weight %	Uranium content %	uranium distribution %	Weight %	Uranium content (ppm)	Uranium distribution %
50	34.3	19	0.3	65.7	3 640	99.7
500	51.9	83	1.9	48.1	4 999	98.1
1 000	60.1	178	4.2	40.0	5 841	95.8
2 000	74.5	426	13.0	25.5	8 354	87.0
3 000	80.6	572	19.0	19.4	10 203	81.0
4 000	84.5	706	24.6	15.5	11 871	75.4
5 000	87.8	847	30.6	12.2	13 906	69.4
10 000	93.8	1 229	47.4	6.2	20 283	52.6

It appears that a slight enrichment occurs in the coarser fractions and also in the fines. The reconstituted raw ore uranium content is 2 052 ppm. The table also indicates that the ore is very hard, the quantity of fine particles is not important. The cumulative results for the +30 mm and -30 mm fractions are:

Size fractions mm	Weight %	Uranium content ppm	Uranium distribution %
+30	64.6	2 312	75.4
-30	35.4	1 4323	24.6
Total ore	100.0	2 052	100.0

These results indicate that two thirds of the total weight and three quarters of the uranium content can be treated by radiometric ore sorting.

4. RADIOMETRIC ORE SORTING TESTS RESULTS

The characteristics of the classes are given in Table II for the three coarser fractions (30–50 mm, 50–80 mm, +80 mm).

TABLE VII. ORE SORTING TEST RESULTS (From raw ore (U = 2052 ppm), weight, uranium content and uranium distribution for the accept, reject and fines < 30 mm fractions)).

Cut off grade		Weight %	Uranium content ppm	Uranium distribution %
50	Accept	42.4	3 640	75.2
	Reject	22.2	19	0.2
	< 30 mm fraction	35.4	1 432	24.6
500	Accept	31.1	4 999	74.0
	Reject	33.5	83	1.4
	< 30 mm fraction	35.4	1 432	24.6
1 000	Accept	25.8	5 841	72.2
	Reject	38.8	178	3.2
	< 30 mm fraction	35.4	1 432	24.6
2 000	Accept	16.5	8 354	65.6
	Reject	48.1	426	9.8
	< 30 mm fraction	35.4	1 432	24.6
3 000	Accept	12.5	10 203	61.1
	Reject	52.1	572	14.3
	< 30 mm fraction	35.4	1 432	24.6
4 000	Accept	10.0	11 871	56.9
	Reject	54.6	706	18.5
	< 30 mm fraction	35.4	1 432	24.6
5 000	Accept	7.9	13 906	52.3
	Reject	56.7	847	23.1
	< 30 mm fraction	35.4	1 432	24.6
10 000	Accept	4.0	20 283	39.7
	Reject	60.6	1 229	35.7
	< 30 mm fraction	35.4	1 432	24.6

According to the results, the following information should be given:

- first, it is possible to eliminate a large quantity of ore without an important uranium loss.
- second, the uranium content is increasing in the accepted fraction according to the detailed results given in Table III for the 30–50 mm class, Table IV for the 50–80 mm class, Table V for the +80 mm class, Table VI for the cumulated fractions. In these tables, for various cut off grades, the weight, uranium content and distribution will be found for the rejected or accepted fractions.

The uranium distribution does not vary much with larger sizes, but the selectivity decreases with the coarser fractions. This is not surprising. The enrichment coefficient in the accepted fractions decreases from 4.4 on the 30–50 mm fraction to 3.5 on the 50–80 mm and to 2.4 on the +80 mm fractions.

In Table VII, the same ore sorting results are given but they are cumulated with the -30 mm fraction. Further tests will be then developed towards other concentration techniques for the -30 mm fraction.

5. CHEMICAL CHARACTERISTICS OF THE ORE AFTER RADIOMETRIC SORTING

The first observations which can already be given are:

	Raw ore %	Ore after ROS %
U	2 052 ppm	8 858 ppm
Fe ²⁺		3.14
F ³⁺	3.5	0.06
S ²⁻	0.42	0.4
Mo	210 ppm	750 ppm

The first observations which can already be given are:

- 1) The uranium enrichment is near 4.
8 858 ppm compared to 2 052 ppm.
- 2) The molybdenum concentration increases according approximately to the same ratio as for uranium.
- 3) The detrimental elements contents of the sorted sample are close to those of the raw ore before radiometric ore sorting.

6. GRAVIMETRIC TESTS CARRIED OUT ON THE PART OF THE ORE NON AMENABLE BY RADIOMETRIC SORTING

The uranium size analysis after mining operations are summarized hereafter.

	Weight %	Uranium %
Part 1 +30 mm	64.6	75.5
Part 2 -30 mm	35.4	24.5

TABLE VIII. RESULTS OF HEAVY LIQUIDS TESTING AT DIFFERENT ORE SIZES

	Size fraction μm	Weight %		Uranium content ppm		Uranium distribution %		Iron content ppm		Iron distribution %	
Light $d < 2.95$	> 50	52.11		64		2.44		3.338		42.6	
Middlings $2.95 < d < 3.29$	> 50	7.76		1 761		9.96		5.873		11.1	
		Magnetic	Non magnetic	Magnetic	Non magnetic	Magnetic	Non magnetic	Magnetic	Non Magnetic	Magnetic	Non magnetic
Heavy $d > 3.29$	> 50	0.20	0.81	3 490	101 811	0.51	60.12	52.234	14.69	2.6	2.9
Fines	< 50	38.92		953		26.97		4.314		40.8	
Raw ore	-	100.00		1 372		100.00		4.103		100.00	

Part 2 was ground (95% minus 500 μm) and tests were carried out with two dense liquors of respective densities 2.85 and 3.29.

The tests produced three fractions:

- a light fraction $d < 2.85$,
- a middle fraction $2.85 < d < 3.29$
- a heavy fraction $d > 3.29$.

The results reported at the Table VIII show that the minus 30 mm fraction is amenable by gravimetric separation. For the part coming from radiometric sorting, the uranium content will be 7 000 ppm. The two fractions coming from gravimetric separation, heavy fraction plus middlings, give an uranium content of 1.7% and an uranium recovery of 70%.

These results were confirmed at an industrial test using the KNELSON concentrator. This robust equipment is used in the gold industry.

7. CONCLUSION

Radiometric ore sorting is a very good technique for EKO REMAJA ore to increase the uranium content. The uranium in the coarser fractions amenable by ROS represents 75% of the total uranium.

The possibility to recover by gravimetric operation a large part of the uranium non amenable by ROS shows that EKO REMAJA ore fits very well to physical or radiometric enrichments.

The largest part of uranium (80–90% according to the cut off grade) can be recovered with an enrichment ratio included between 3 and 4.

The uranium extraction of EKO REMAJA is the part of the KALAN project on which we have progressed quickly.

A large part of the data necessary for the feasibility study are available. We have now to define and make a financial analysis of the necessary improvement of Lemajung pilot plant.

The same equipments with minor modifications will allow to recover a part of the uranium linked with the rare earths in RIRANG boulders.

Rare earths stay in the solid residue but with a high level of radioactivity. Today, it seems difficult, maybe impossible, to sell rare earths concentrates containing such a level of radioactivity without complementary treatment.

RESEARCH ON THE COUNTER-CURRENT HEAP LEACHING PROCESS FOR TREATING COMMON GRADE URANIUM ORES IN CHINA AND THEIR RECENT DEVELOPMENTS

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Abstract

With the continuous increase of the installed capacity of nuclear power stations, the depressed situation of the uranium industry cannot continue indefinitely. The development of uranium ore processing technology will show its lasting importance. Around the world more and more attention has been given to the underground leaching and heap leaching technologies. The uranium ore grade treated by heap leaching is gradually increasing, and the trend to replace agitation leaching is becoming more and more evident. Therefore it is necessary to do considerable commercial development work for its application in treating the common grade uranium ores to obtain satisfactory technic-economic performances. So the future is optimistic. In this paper recent research work and development are described on treating uranium ores with a grade of 0.3-0.4% U_3O_8 by applying the counter-current heap leaching process. At the present time, the author considers that, shortening the heap leaching period, and programming the leaching operations are the main objectives in the development of heap leaching technology for the recovery of uranium. The above goal can be achieved by bacterial heap leaching using concentrated sulfuric acid mixing and curing-ferric sulphate trickling. Being an intensified process, the counter-current operation characterized by using bacterial-oxidized regeneration solution and ore heaps in counter currents, can shorten the heap leaching period, and increase the uranium recovery rate. The recovery of uranium ore located in metamorphosed igneous rock type deposits can reach 96% or more when the ore is treated as follow: the ore is crushed up to 6 mm, the uranium grade is 0.3-0.4% U_3O_8 , the solution with ferrous ions is oxidized by bacteria, the $\Sigma Fe > 16$ g/L, the pH is 1.2-1.6, the leaching period is 60 d, the acid consumption is 46 kg/t, there are 4 count-current stages. The U_3O_8 content in the residue can be decreased to 0.02%. Acid mixing is a granulation process of multi-sized materials with concentrated acid, which can fairly avoid the size segregation, improve the kinetics of the leaching process and considerably shorten the heap leaching period. The pilot-plant test results of concentrated sulfuric acid, curing-ferric sulphate, count-current heap leaching, showed that acid consumption (40 kg/L) can also reduce the uranium content in the residue to 0.02% U_3O_8 . At the same time the counter-current operation can bring the leaching solution with higher uranium concentration, and fully develop the features of the better selectivity of heap leaching. It provides suitable conditions and environments for selection of extraction technology, survival and reproduction of bacteria and increase activation. Programmed counter-current heap leaching operation instead of single heap leaching operation can make the mill to produce high quality uranium concentrates. This is an important development of heap leaching technology for commercialization.

1. INTRODUCTION

With the development of nuclear energy, uranium has become an important source of energy resource in the world. Nuclear energy is regarded as a safe, clean and economic energy resource. The total nuclear-electric generating capacity represents up to 17% of the world electrical energy. The recession of the uranium industry caused by the oversupply in the market will certainly will not last forever. The technology of uranium ore processing will become more important with the increase of the total installed nuclear-electric generating capacity. The main tasks at present in the uranium ore processing technology are: to increase the leaching rate of uranium, to cut down the cost of processing, to produce high quality concentrates and to minimize waste. For those demands, in situ leaching and heap leaching technology have received much attention by scientists, engineers and production units.

Progress and improvement in the heap leaching technique have been made around the world in recent years. The grade of ore processed in heap leaching is gradually raised. There has been a tendency to replace the agitation leaching process by heap leaching. The economic benefits of heap leaching depend mainly on the ore grade, production scale and leaching rate. The larger the scale of production, the better the economic benefit as compared with the conventional process. The main

purpose of the heap leaching process is the replacement of the agitation leaching process. It will increase the leaching rate and shorten the leaching period, it will help to programme heap leaching operation and finally realize the industrial scale production.

In this paper the research and development of heap leaching of uranium ore in China is described, especially the counter-current heap leaching of larger granular uranium ore, bacterial heap leaching, concentrated sulfuric acid mixing and ferric sulphate curing processes.

2. COUNTER-CURRENT HEAP LEACHING BY USING BACTERIAL-OXIDIZED REGENERATION SOLUTION

It is well known that counter-current leaching is an effective method which can intensify the leaching process, enhance the leaching selectivity and reduce the reagent consumption. Counter-current heap leaching is certainly beneficial for intensifying the leaching operation, shorten the leaching period and increase the leaching efficiency. When the leaching conditions are determined, the leaching reagent flows against the ore pile direction and the leaching of uranium gradually takes place. The following are the conditions of the four stages of the counter-current heap leaching operation. In this experiment, the leach rates of uranium were measured to examine the efficiency of the leach process.

Ore Size	~6 mm
Ore grade	0.36–0.4% U_3O_8
Compositions of solution of leaching reactant	
ΣFe	7–15 g/L
$[Fe^{3+}]/[Fe^{2+}]$	$5/2 \pm 0.1$
pH	1.2–1.6
Oxidation potential	450–680 mV
Temperature	ambient temperature (12–28°C)
Sprinkling rate of lixiviant	80–170 L/h.m ²
Yield of leach liquor	0.6–0.8 m ³ /t ore

Compared with separate heap leaching of uranium ore, mixture of uranium ore and pyrolusite by sulfuric acid solution, the counter-current heap leaching of the uranium ore by the solution from the bacterial-oxidized regeneration has the general advantage of normal heap leaching process. It can also reduce by 25–40% the sulfuric acid consumption, and increase by 50% the leach rate of uranium. In addition, it is not necessary to add the pyrolusite to the leaching process.

3. THE OXIDATION ACTION OF BACTERIA IN THE COUNTER-CURRENT HEAP LEACHING PROCESS

The bacteria *Thiobacillus ferro-oxidans* was self-cultured by the pyrite in the ore and Fe^{2+} ion in the solution and introduced into the heap leaching process. During the counter-current operation, the leaching solution supplied the nutrients for the bacteria and the ore pile acted as a support for the bacteria. A considerable amount of bacteria absorbed on the surface of the ore formed a bacterial film. When sufficient air was supplied, parts of pyrite, the tetravalent uranium in the ore and ferrous ions in the leaching solution were oxidized under the direct or indirect actions of bacteria which maintain the necessary oxidation potential in the leach process.

The oxidation potential in the leaching solution of uranium was under 450 mV during the first ten days of operation. When the leach was continued, the oxidation potential increased over 500 mV with the bacterial effect. When the balance of the four stages of counter-current leach was reached, the oxidation potential of the fourth stage leaching solution was below 480 mV. The oxidation potential of the other three stages solutions were between 490–660 mV. In this situation, most of the

ferrous ions were oxidized and provided the satisfactory oxidation conditions for heap leaching. The leach rate of uranium was increased and the leaching period was shortened. Multi-pile counter-current leaching operation in replacement of single-pile leaching can obtain a leaching solution containing high concentration of uranium. Taking the advantage of good selectivity of heap leaching, the optimum conditions of extraction technology were determined and fully used in the leaching operation of uranium. Programmed counter-current heap leaching operation instead of a single heap leaching operation, result in producing high quality and stable uranium concentrates. Heap leaching instead of agitation leaching process was an important development in the treatment of common grade uranium ore.

Increasing the oxidizing rate of ferrous ion and shortening the oxidizing time should be considered when ensuring the necessary conditions of regeneration and oxidation of the leaching solution. In addition of selecting the species of bacteria, improving its adaptability to the environment, a filler should be put into the leaching cells in order to maximize the activated bacteria absorbed on the filler as support to form a large group of bacteria area. It is essential to fully contact the leach solution, for the reduction of the growing incubation time of bacteria and increase the oxidizing rate of ferrous ions by bacteria.

4. ACID MIXING AND CURING-FERRIC SULPHATE TRICKLING HEAP LEACHING

Acid mixing process is an agglomerating process of concentrated acid to the different size of ores. To make the counter-current leach a stable operation, it is beneficial to shorten the leach operating period. It prevents the effect of grade segregation on the leaching process, and improve the kinetics characteristics of the leach process e.g. intensifying the dissolution of uranium into the solution. It also restrains the leaching out of the acid-consumption minerals and improves the selectivity of the leaching.

After mixing concentrated acid and allowing ten days for curing, the ore piles is sprayed with the ferric solution to leach uranium. Miniature spray system with upper double port, was used in the leach process. Four stages of counter-current heap leach were employed in the pilot-plant test. One pile has about 3.5 tonnes of ores. Each stage took ten days to leach. The total leaching period is forty days.

Ore size	-6 mm
Ore grade	0.37% U_3O_8
Ore moisture content	< 6%
Acid cured consumption (by ore weight)	4%
Actual acid consumption	2.0%
Acid solution	0.1-0.12 m ³ /t. ore
Maturing time (ambient temperature)	10 days
Ore-heap height	4.0 m
Leach liquor yield	0.6-0.7 m ³ /t.ore
Leach rate of uranium	93-95%

The uranium was recovered from leaching solution by either two steps precipitation or trialkyl amine extraction. After precipitation or extraction, the solution or raffinate was oxidized by bacteria or pyrolusite, and then return into the preparation section of leaching reagent. The processed solution is recirculated and no waste water is discharged to the environment. The experiment ran about 30 days after the test balance was reached. During the experiment, the accumulated impurity had no bad effect on the leach process. The product of ADU meets the quality standard.

5. CONCLUSION

In view of the trend of development and application of the heap leaching technology, the following questions were proposed for discussion.

5.1. The leach rate in heap leaching

For maximizing the leach rate of uranium, shortening the leaching time is an important factor for achieving the industrial and engineering scale production and expanding application of heap leaching. It should be pointed out that heap leaching has the following properties:

- The grain size of the ore has a great influence on the leach rate. When the ore is highly compacted and hard and the uranium is uniformly distributed it is necessary to grind the ore. When the ore is loosely consolidated and porous the grain size has little effect on the leach rate.
- The uranium leaching process in heap leaching can be divided into two phases. The uranium in the fine fraction of the ground ore was leached out first, then the leaching took place by the diffusion into the pore of the ore. The later process is much slower than the former process. A better leach rate is obtained by means of extended leaching time in usual heap leaching. As discussed before, the technique used in this paper can shorten the leach time for satisfactory leach rate.

5.2. The limit of ore grade in the heap leach

Considering the mine and the mining cost, the industrial experiment indicated that when the uranium content in the ore is about 0.07–0.08% U_3O_8 , the heap leach is at the balance state of lost and benefit. The leach rate is increased as the ore grade increases. Correspondingly, a high leach rate brings about high economic benefit. For the grade of ore increase, the reduction of one percent of leach rate can cause serious losses in the economic cost. Therefore, for the high grade ore, the uranium leach rate should be over 99%. When the grade of ore is between 0.1–0.3%, the normal leach rate can be reached, which is about 90–95%. Thus, the leach rate cannot be below 95% treatment for ore content of 0.3% U_3O_8 . In this paper, the technique used in the heap leach process could obtain a leach rate of about 96%.

5.3. The treatment of the heap leaching solution

With the increase of ore grade in the heap leaching, and application of counter-current operation, the uranium concentration in the leaching solution was increased naturally. This could be obtained with the selection of the uranium extraction technology such as chemical precipitation and solvent extraction. The ore property, composition of leaching solution and technology demand, should be considered when selecting the process method. Test results showed that under proper control of $CaSO_4$ and Cl^- etc., impurity elements, ion exchange system can process the solution with uranium concentration as U_3O_8 2.0 g/L. The uranium content in effluent is under 5.0 mg/L and the capacity of resin is over 40.0 g/L. The yield of ADU product can meet the quality standard. This indicated that ion exchange technology can be used to treat the concentrated solution of uranium from heap leaching process. The method is feasible. So that the uranium from the heaping leaching solution can be recovered by conventional extraction technology.

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